TRANSACTIONS

1949

A. I. M. E.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

Volume 181

American Institute of Mining, Metallurgical and Petroleum Engineers.

METAL MINING, INDUSTRIAL MINERALS, MINERAL INDUSTRY EDUCATION, AND GEOPHYSICS

1949

PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD AT CHICAGO, FEB. 25–28, 1946; LOS ANGELES, OCT. 25, 1946; OCT. 24, 1947; TUCSON, OCT. 28–30, 1946; NEW YORK, MARCH 17–19, 1947; FEB. 15–19, 1948; DENVER, OCT. 1, 1947; ATLANTA, OCT. 8–10, 1947.

PUBLISHED BY THE INSTITUTE
AT THE OFFICE OF THE SECRETARY
29 WEST 39TH STREET
NEW YORK 18, N. Y.

PRINTED IN THE UNITED STATES OF AMERICA

THE MAPLE PRESS COMPANY, YORK, PA.

FOREWORD

In this volume will be found much of interest to the man whose job it is to find ore and get it out of the ground. Three of the papers are on geophysics and four on exploratory drilling. By far the larger number of papers, however, are on various features of modern mining practice. The subjects include diamond-drill blasthole stoping, mechanization, jumbo drilling machines, shaped explosive charges, Diesel power underground, mining oil shale, and handling acid mine water. Ground support, or the science of barodynamics, is the subject of six papers. Ever-deeper mines require more study of this subject. The symposium on grouting will be of especial interest to those who have water problems. Two papers are devoted to health and safety in mining operations, a subject about which too much study cannot be given as long as mining is the hazardous industry that it has so far been.

Closely allied to metal mining is the nonmetallic industry, so eight papers of the Industrial Minerals Division of the Institute are included in this volume, four of which discuss foreign operations. Two papers on mineral industry education complete the volume.

With the exception of the 1947 volume of the Transactions (No. 173) devoted to nonmetallic minerals, no volumes on the subjects covered herein have appeared since Vol. 163 (1945) on mining practice, and Vol. 164 (1945) on geophysics. This volume includes papers on the subject matter that were published in the "Technologys" up to the end of 1948. An occasional paper of interest to more than one group that might have been included here has been omitted to obviate duplicate publication.

Hereafter, all technical papers in the fields of exploration; geology; metal, nonmetallic, and coal mining; and beneficiation will be published combined in one volume of 8½ by 11½ inch size. These volumes will appear annually, and should be available within two or three months after the end of the year during which they have been published in the monthly journal, "Mining Engineering." An occasional paper on mineral industry education and economics will also be included.

Though in a larger size, which will not match the earlier volumes issued during the last three quarters of a century, much more prompt availability of the bound volumes will be possible.

All papers published by the AIME have been reviewed by several competent critics, and many have been extensively revised by the authors following first submission and presentation. No special invitation is required to submit a paper to the Institute, nor need an author be a member in order to have his paper considered for publication. If it records worthwhile achievement in the mineral technology field anywhere in the world it will be welcomed.

New York, N. Y.
May 27, 1949

E. H. Robie
Secretary

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The Application of Oil-well Surveying Instruments and Technical Services in the Mining Industry

By G. L. Kothny*

(Chicago Meeting, February 1946)

Developments of well-surveying instruments, coring and core orientation, were in an advanced state when drilling for oil began—these developments actually originated with the mining industry.[1]

Surveying of oil wells was not generally used in the United States until Alexander Anderson[2] disclosed that the majority of wells drilled with rotary tools were crooked, with average horizontal drifts of about 516 ft. for holes 5000 ft. deep and about 793 ft. for holes 6000 ft. deep.

This disclosure brought about the development of many additional instruments for surveying oil wells during the progress of drilling and after their completion. Surveying of boreholes, during the process of drilling at regular depth intervals, became a standard practice in the petroleum industry. Some producing states even established laws making check surveys mandatory, during drilling and under certain conditions also after completion.

The surveying of boreholes also brought about changes and improvements in drilling practices. It provides information for the guidance of the hole while drilling, and for plotting the course as the work progresses, enabling the driller to take corrective measures immediately and with the least expense.

The reasons for drilling straight holes are too well recognized to warrant any full discussion of them here. Straight holes are less costly to drill, because they reach the object on the shortest course. Table 1 shows the horizontal drift and loss in vertical depth for various angles of inclination.

Table 1.—*Horizontal Drift and Loss in Vertical Depth per 100 Feet of Measured Depth*

Feet

Angle	Drift	Depth Loss	Angle	Drift	Depth Loss
½	0.87	0.00	11	19.08	1.84
1	1.75	0.02	12	20.79	2.19
1½	2.62	0.03	13	22.50	2.56
2	3.49	0.06	14	24.19	2.97
2½	4.36	0.10	15	25.88	3.41
3	5.23	0.14	16	27.56	3.87
3½	6.10	0.19	17	29.24	4.37
4	6.98	0.24	18	30.90	4.89
4½	7.85	0.31	19	32.56	5.45
5	8.72	0.38	20	34.20	6.03
6	10.45	0.55	22	37.46	7.28
7	12.19	0.75	24	40.67	8.65
8	13.92	0.97	26	43.84	10.12
9	15.64	1.23	28	46.95	11.71
10	17.36	1.52	30	50.00	13.40

It is not surprising, therefore that many drilling contracts for oil wells today limit the drilling contractor to a maximum inclination off the vertical of 2° or 3°. In the mining industry the reasons for drilling straight holes are still more serious. A crooked hole may miss an ore body underground or provide incorrect information regarding an ore body.

Types of Surveying Instruments

The surveying instruments used in oil-well drilling may be classed in two types:

1. Inclinometers, recording inclination only.

Manuscript received at the office of the Institute Nov. 3, 1944; revised Sept. 21, 1945. Listed for the New York Meeting, February 1945, which was canceled. Issued as TP 1964 in Mining Technology, January 1946.

* Vice-President, Sperry-Sun Well Surveying Co., Philadelphia, Pennsylvania.

[1] References are at the end of the paper.

2. Directional well-surveying instruments, recording inclination and the direction of the deviation.

If they produce single records, they are called Single Shot instruments and if they produce multiple records, they are named Multishot instruments. The records are produced by liquid stains, mechanical impression, photographs, or by electrochemical means.

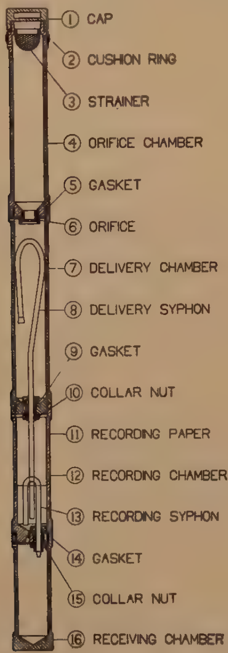

FIG. 1.—CROSS SECTION THROUGH SYFO.

Directional single-shot and multishot instrument records are mostly made photographically. Records of direction are made by the use of a magnetic compass or by a gyroscopic directional indicator, or by measuring the rotation of the drill pipe, through sighting. The last method, however, is now seldom used.

The recording usually is controlled by a timing apparatus (clock, liquid orifice) or manually from the surface, or automatically by the instrument, when it comes to rest.

Requirements of Instruments

Instruments used for surveying boreholes must answer the following requirements: They must be simple, easy to operate, produce true and accurate records and must sustain such accuracy in service. They must be self-checking; i.e., it must not be possible for the instrument to produce a false record without the operator of the instrument recognizing such a false record. Most instruments, except those recording by mechanical impression, are self-checking. Records produced by the instrument should be made in the shortest time possible and should be clear and easy to interpret.

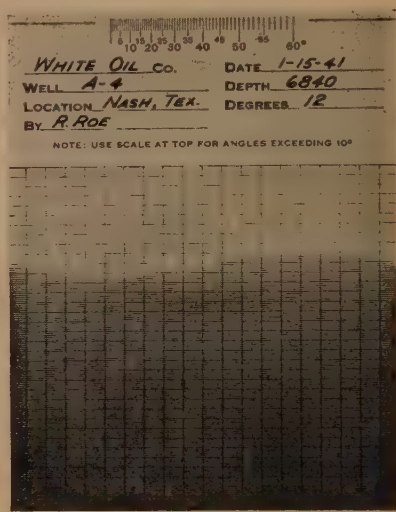

FIG. 2.—SYFO RECORDING CHART.

Multishot instruments should have ample recording capacity to allow the making of two surveys on one round trip into the hole, one on the in-run and the other on the out-run.

Instruments in Use

Inclinometers

The oldest inclinometer of the liquid recording type, the acid bottle, is now seldom used, since its recording is not

sufficiently accurate and is too slow for modern drilling practice. It has been replaced by a syphonic instrument (Fig. 1), operating on the same principle as the acid bottle but using a harmless dye and a paper recording chart (Fig. 2).

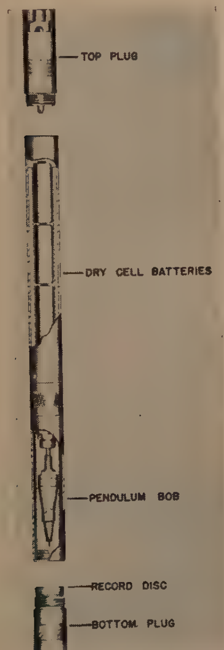

FIG. 3.—CROSS SECTION ELECTROCHEMICAL INSTRUMENT.

An inclinometer of electrochemical recording type is shown in Fig. 3. A chemically treated paper disk of dark blue color with printed circles indicating the degrees of inclination is continuously in contact with a plumb-bob pendulum through a floating platinum pin. Three pencil-type dry cell batteries furnish a weak electric current, which, when the instrument is put into the protective casing and is held in a nearly vertical position, flows continuously through the pendulum and disk. When the pendulum is at complete rest for a minute or more, the electric current causes a chemical action and changes the color of the recording disk, producing an easily readable record in the form of a white round dot (Fig. 4). The size of this dot increases with the length of time of the resting

FIG. 4.—RECORD OF ELECTROCHEMICAL INSTRUMENT.

period. This feature makes it possible to obtain several records during one round trip in the hole, inasmuch as the depth at which each record is taken can be identified by varying the resting periods and the corresponding sizes of the dot.

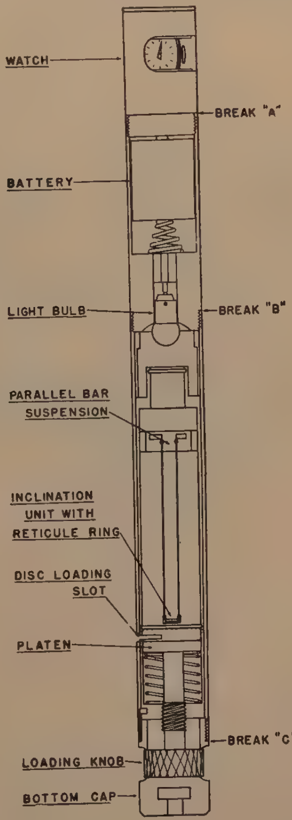

FIG. 5.—H-K INCLINOMETER CROSS SECTION.

During the lowering or raising of the instrument, no records are produced; during these periods the pendulum is always in motion and the amount of

FIG. 6.—H-K INCLINOMETER RECORD.

electric current passing through the disk is insufficient to produce the chemical action.

An inclinometer of the photographic recording type is illustrated in Fig. 5. A pendulum suspending a fine cross hair immediately above a glass disk having etched concentric circles representing degrees of inclination is photographed, together with the shadows of the concentric circles on a sensitized paper disk (Fig. 6) inserted by means of a daylight loader (Fig. 7) immediately below the glass disk. An electric bulb, receiving electric current when the timing clock closes the switch, provides the illumination. The recording disk is removed from the instrument into a daylight developing tank (Fig. 8), in which the developing and fixing are done on the derrick floor in 4 to 5 minutes.

An inclinometer of the mechanical type is partly shown in Fig. 9. The record disk holder, containing a printed record disk, is released at a time set beforehand and moves toward a compound pendulum, the sharp end of which perforates the recording disk. This type of inclinometer is not self-checking.

Directional Well-surveying Instruments

Photographic means are used mostly in these instruments for recording the inclination of a cross-hair type of pendulum and the bearings of a magnetic compass or a gyroscopic directional indicator.

Fig. 10 illustrates a directional single-shot instrument of this type. The lower part of this instrument is similar to that of the inclinometer shown in Fig. 5. Between the light bulb and the inclination unit there is arranged a lens chamber and a floating magnetic compass, which is light transparent. The compass has a circumferential bearing scale, the record of which is superimposed upon the inclination record on the sensitized paper disk (Fig. 11), producing a combined record.

The timing, loading of the instrument and the developing of the record are similar to that of the photographic type of inclinometer.

Multishot directional surveying instruments use a somewhat similar arrangement, except that a moving picture camera is used for photographing the records on an 8-mm. or a 16-mm. film. Multishot

FIG. 7.—H-K DAYLIGHT LOADER.

FIG. 8.—H-K DAYLIGHT DEVELOPING TANK.

instruments are self-contained and operate electrically, by current supplied from dry cell batteries or manually through electric current supplied by electric cable from the derrick floor.

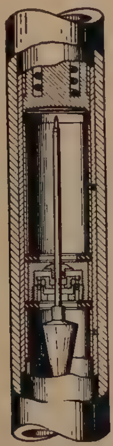

FIG. 9.—MECHANICAL INCLINOMETER.

These instruments are of too large a diameter to be used in diamond-drilled holes, therefore their description is omitted.

METHODS OF OPERATION

Methods of operation of inclinometers and directional single-shot instruments are described on pages 79 and 80 of reference 1.

FREQUENCY OF RECORDING

No definite rule with regard to the use of either inclinometers or directional single-shot instruments has been established. Inclinometers are considered necessary instruments by an efficient drilling organization, and are used daily, or even more frequently if drilling progresses rapidly. Directional single-shot instruments come into play more on wildcat wells or directionally drilled holes.

The frequency of surveys depends on conditions. In a new field or in strange areas the taking of a record every 50 ft. would not be too often. As information is gained on the area, it might be safe to decrease the number of records to one every 200 ft., with additional tests at selected points immediately under formation that may particularly affect the devia-

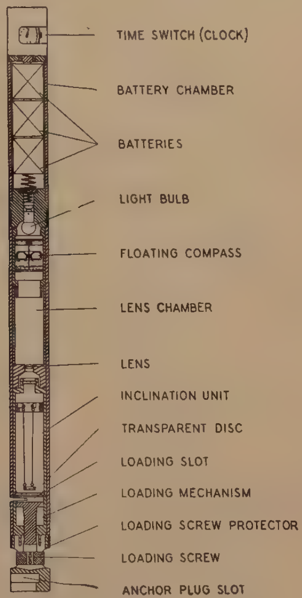

FIG. 10.—H-K SMALL DIRECTIONAL SINGLE SHOT.

FIG. 11.—H-K SMALL DIRECTIONAL SINGLE-SHOT RECORD.

tion. Where it is known that dipping formations are causing crooked holes, it might be well to increase the number of records.

BENEFITS OBTAINED

The records obtained provide the necessary information for formulating the technique and drilling methods to be used.

If the record indicates the tendency of the hole to deviate, corrective measures

should be taken at once. Straightening of rotary drilled holes that are beginning to deviate is usually done by reducing the weight a few points and increasing the drilling speed. Experience usually dictates the proper use of weight and proper correlation of bit weight with the geological formation. Control is accomplished by use of accurate instruments for indicating the weight carried on the bit, the speed of rotation and the mud pressure.

Experience has demonstrated that the use of surveying instruments has eliminated crooked holes and has reduced drilling costs. The cost of surveying, cost of instrument rental and cost of rig time for surveying are small when compared with the benefits resulting from their use.

Instruments Used by the Mining Industry

Most exploratory holes are drilled with the diamond drill. Diamond drillers at present use mostly the acid bottle, the Carlson compass, the Maas compass, and the Radio-Lite instrument, which have been described in detail in various publications.

The time required for obtaining records with these four instruments is far greater than that required by the inclinometers and single-shot instruments just described, and the records obtained are not as accurate as may be desired.

Inclinometers and directional single-shot instruments of smaller diameters for use in diamond-drilled holes can be developed if the mining drilling industry would show interest in their continuous use.

Adaptation of Instruments and Drilling Methods Used by the Petroleum Industry

Small single-shot instruments having $1\frac{11}{16}$-in. o.d. protective casings have been successfully used by seismographic crews in diamond-drilled holes to depths of 500 ft. to determine the correct location of the charge at the bottom of the hole and to obtain more accurate information when interpreting the seismographic records. These instruments could be used by the mining industry in the $1\frac{57}{64}$-in. holes.

Inclinometers of the electrochemical type with protective casings of $1\frac{5}{16}$ in. o.d. are also available and could be used in size EX and size AX diamond-drilled holes.

Straight-hole drilling methods should be applied, to maintain the inclination of the hole within 2° off the vertical. Drilling contracts should specify limits for the vertical deviation and should require the taking of records at regular depth intervals, depending upon the depth of the hole and the type of geological formation traversed.

Recognizing the many advantages gained in the petroleum industry from surveying boreholes and from using available measures for promoting vertical holes, it should be possible to apply these developments to diamond-drilled holes and reap the similar benefits. However, when methods and ideas are transferred from one branch of industry to another, they require more than the physical alteration necessary to fit new operating conditions. It is also necessary to consider thoroughly their underlying principles and the reasons for their use, if they are to serve their full purpose.

There is no doubt that the combined efforts of mining engineer, driller and instrument manufacturer could produce instruments and methods applicable to all the various drilling operations of the mining industry.

It brings forth the question: Why has the mining industry, from which well surveying originated, not yet adopted the techniques that have proved so beneficial to the petroleum industry? One answer may be that the benefits resulting from

straight-hole drilling methods have not been sufficiently appreciated by the drillers in the mining industry to assure a general application of these techniques and a wide use of the instruments. Limited applications, as well as outright purchases of instruments, which mining engineers seem to prefer, furnish little encouragement to the instrument manufacturer to undertake the development of new types or sizes. The cost of development, tooling up and maintaining replacement parts on hand are very high and justify a reasonable assurance of commensurate returns, which cannot be expected from occasional sales of instruments. If development costs are apportioned to a limited number of instruments sold, the purchasing price of the instrument will be very high, and this will not stimulate sales.

When a company or the manufacturer of the instrument specializes in giving technical rental services, the costs of development are apportioned among a large number of instruments, and are paid for by all the renters in accordance with the use they make of the instrument. Renters also have the benefit of improvements made from time to time in the design or operation of the instrument, which a purchaser could obtain only at additional costs.

Core Orientation

The practice of taking cores from boreholes has become so common that their value is well recognized. However, a core sample does not yield all possible information, unless the exact location from which it came is also known and unless the sample is oriented.

A directional survey of the part of the hole from which the sample is taken is a necessity for the correct determination of the direction as well as the angle of the dip of formations.

The first attempts to orient cores date back to the year 1854. Since that time, apparatus has been developed that can be grouped in the following classes:

1. Orienting the core barrel by drill-stem orientation.

2. Taking a photographic record with a directional single-shot instrument, which is in a definite and known location to the core sample or core barrel.[3]

3. Measuring electrically the dip of the strata in the open hole by means of anisotropy (exhibiting different properties when tested in different directions).[4]

4. Finding the direction of the dip of the core sample by determining the retained earth polarity by a very sensitive magnetometer.[5]

The first two methods require considerable rig time and the work is done with the risk that the sample may not contain good bedding planes. The third method requires larger holes, as there are physical limitations to the size of instrument used for this purpose.

The fourth method makes it possible to orient samples without interfering with the drilling work and to pick out from core trays samples that show desirable bedding planes. The orientation can be made at any time provided the samples have not been left in the vicinity of any direct or alternating current field or in the vicinity of any permanent magnetic field.

Method 4 lends itself best to the orientation of diamond-drilled cores. The core samples selected for this method should show visible indications of bedding planes and as soon as taken from the core barrel should be marked "top" and "bottom." The samples selected should contain mineral grains, distributed throughout, such as, magnetite, pyrrhotite, ilmenite, chromite, almandine, and glauconite. Monazite, staurolite and tourmaline are weakly magnetic, but sometimes can be used with this method. Quartz, feldspars, zircon, kyanite and spinel are practically nonmagnetic and cannot be used with this

method. Also, has it been found that pure limestone, diatomites, anhydrites, and dolomites show no polarity whatever.

A detailed description of method 4 is contained in a paper by Roberts and Webb.[5] This method of core orientation makes it possible to obtain results without loss of rig time and also permits the orientation of old cores, provided these have been kept away from magnetic and electric fields while in storage. It offers many possibilities to the mining industry. The direction and the angle of the dip of ore bodies can be determined by the orientation of core samples from one hole, in lieu of diamond-drilling a large number of holes.

Cores from ore bodies have shown very strong polarization and it has been possible to orient cores ground to ¾-in. diameter.

Conclusion

The history of the use of well-surveying instruments and the changes such use brought about in the drilling operations of the petroleum industries indicate that well surveying has enabled the petroleum industry not only to solve many of the difficulties encountered in drilling operations, but also has very much increased the efficiency of drilling wells.

Similar results could be achieved by the mining drilling industry by the continuous use of surveying instruments described herein. Concerted efforts by the mining engineer, geologist, driller and instrument supplier, will make possible the creation of instruments of smaller sizes for use in connection with diamond-drilled holes.

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5. D. C. Roberts and E. R. Webb: Polar Core Orientation. Paper presented at Technical Meetings during *Oil World* Exposition at Houston, Texas, April 24–29, 1939.

Description of Tro-Pari Borehole Surveying Instrument

By Charles Trotter*

(New York Meeting, February 1948)

An instrument has recently been developed for determining the inclination and azimuthal direction of boreholes. This instrument is known as the Tro-Pari Instrument and is the invention of C. Trotter and G. Pajari of Sudbury, Ontario.

Description

The instrument is characterized by a unit mounted in gimbals and provided with a clockwork mechanism to clamp a compass to indicate azimuthal direction and simultaneously to clamp the unit in its plumb position to indicate inclination (Fig 1).

By means of a timing ring suitably calibrated in 5-min divisions, the unit may be set to lock after a lapse of time sufficient to allow the placing of the instrument at the desired point in the drill hole where readings are to be taken. The maximum time lapse obtainable with the instrument designed for use in *EX* or *AX* size holes is 1 hr 30 min which is considered to be ample time to lower to the greatest depths that may be drilled with this size of equipment. Larger sizes could be developed for larger diameter holes such as *BX* and *NX* sizes or the larger oil-well sizes and a correspondingly greater time lapse could be provided for. Up to the present time only the *EX-AX* size has been built.

Operation

Assuming a borehole is to be surveyed at 1000-ft depth and assuming it will take 30 min to lower the instrument to this depth, the timing ring is rotated through seven divisions which represent a time lapse of 35 min (5-min leeway) and is then returned to its zero position. This operation imparts energy to a spring which drives the clockwork mechanism. Simultaneously the compass needle is freed to swing to its magnetic north-south position and the encased mechanism is freed to hang plumb within a semicircular portion of the frame which is graduated in degrees after the fashion of a protractor. This semicircular protractor portion has a *V*-shaped notch at each degree division of its inner periphery.

The instrument, now in its free position, is inserted in a bronze container made to suit the size of hole to be surveyed (Fig 2). Soft rubber washers are provided within the container to prevent undue shock. The container is connected to the drill rods with three 5-ft lengths of brass or aluminum alloy rods separating it from the steel rods in order to prevent distortion of the magnetic field. The assembly is then lowered to the desired point where readings are to be taken and is allowed to remain at rest until a period of time has elapsed slightly in excess of the time for which the timing ring was set. In the example cited above, when the 35 min has elapsed the compass needle is locked in its north-south position and the protractor locking pin is extruded and locks in the *V* notch opposite its plumb position.

The device may now be removed from the borehole and it will remain in this locked position in spite of jarring or shak-

Manuscript received at the office of the Institute Sept. 16, 1947. Issued as TP 2300 in Mining Technology, March 1948.

* Mechanical Engineer, Trotter-Pajari Instruments, Sudbury, Ont., Canada.

ing, and only becomes unlocked when the timing ring is again moved to take the next reading. From the locked instrument readings are observed for dip and azimuth. With

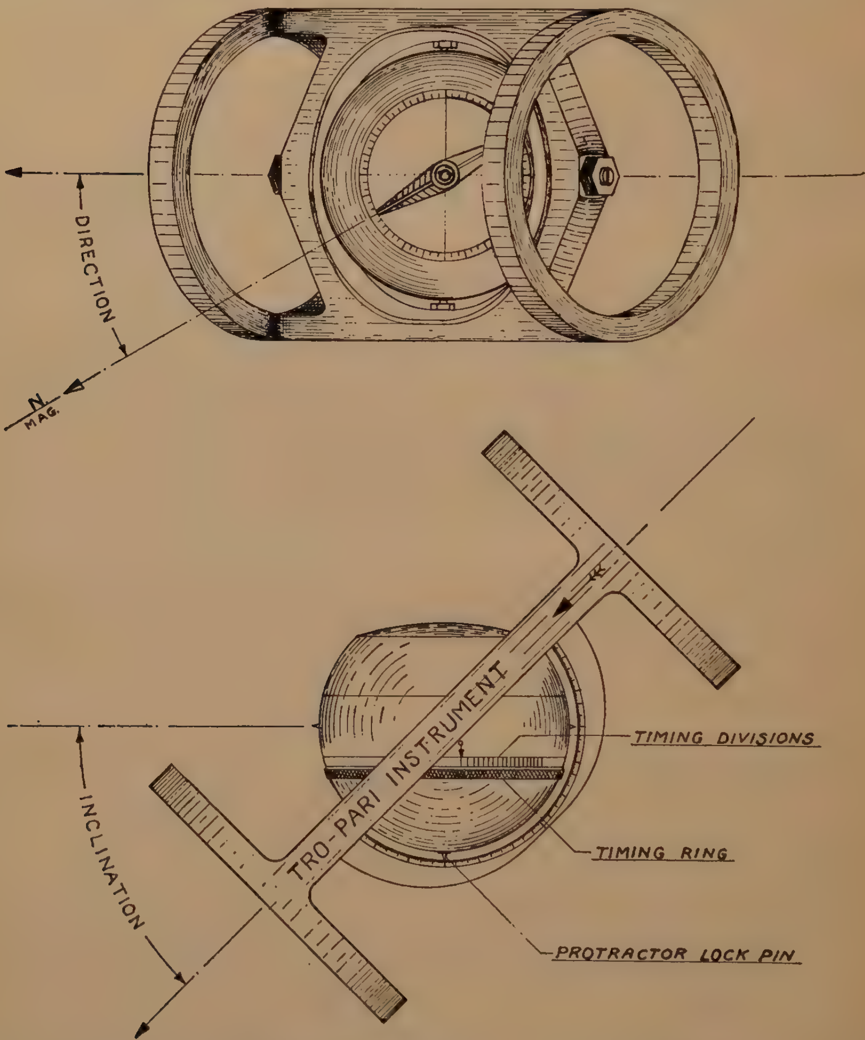

FIG 1—THE TRO-PARI INSTRUMENT FOR DETERMINING DIRECTION AND INCLINATION OF DRILL HOLES.

the aid of a magnifying glass, readings are obtained for both dip and azimuth to within half a degree. There is no necessity for speed in withdrawing the instrument from the borehole since it remains in its locked position indefinitely.

RESULTS

Within the last year, several interesting surveys have been made with this instrument and of these, the deepest hole survey ing job was completed by Pickands, Mather and Co. at their Palms-Anvil mine at Ironwood, Mich., under the direction of their District Engineer, Mr. H. W. Johnson.

Table 1 shows the results of this survey and the corresponding acid test readings.

The following is an extract from Mr. Johnson's report on the above survey:

Check readings were made at various depths and the results were very satisfactory for bearings per eight hours were made from 200′ and 1200′, nine readings from 1200′ to 2700′, five readings from 2700′ to 3000′ and four readings from 3000′ to 3450′.

The work was in charge of a mining engineer,

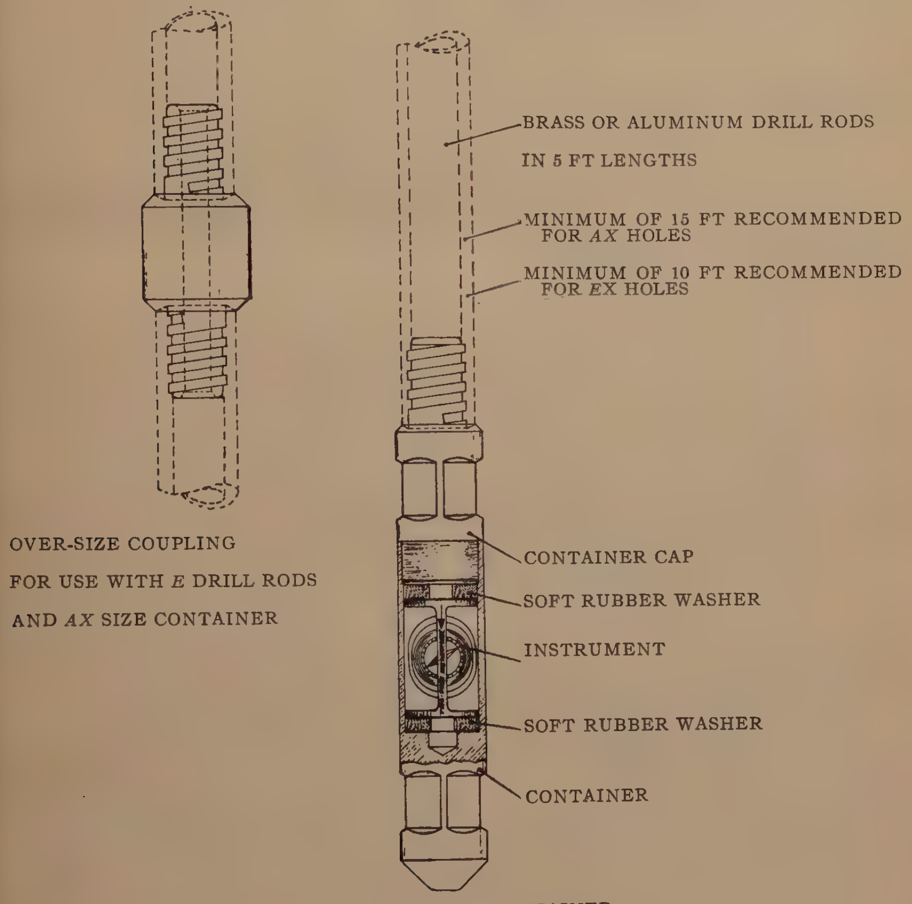

FIG 2—TRO-PARI INSTRUMENT READY FOR USE.

ing and dip, never varying more than a degree in bearing and less than one half degree for dip.

A ten-foot brass rod and the Trotter-Pajari instrument in a brass clinometer were lowered on a steel wire connected to a Halliburton Drill Hole Wire Reel complete with a measuring device. The lowering time to 3500 ft was 15 minutes with 10 additional minutes allowed for the locking device to set. At this depth it took 1½ hours to reel up by hand crank. Twelve readas it is too delicate an instrument to be handled by the drill crew: The crew consisted of the drill runner, drill helper, engineer and his helper.

Under the direction of Mr. Alan C. Lee, Mining Engineer of Noranda, Quebec, surveys were completed on several drill holes at Pitt Gold Mining Company's property in Duparquet Township using a Tro-Pari

instrument. A typical example of this work is given in the results of a survey of Hole No. 70 which was started at S20E and a dip of 85 degrees.

Since nearly all the holes drilled on this property are strongly deflected westward, it is important to have as accurate a survey of them as possible.

TABLE 1—*Results of Pickands Mather and Company Survey*

	Acid Test	Tro-Pari Test	
Depth, Ft	Dip, Deg-Min	Dip, Deg	Bearing Deg-Min
0	59-00	59	Due North
200	61-15	60	N 2-45 W
400	61-20	62	N 12-45 W
600	61-25	62	N 13-15 W
800	61-15	63	N 10-15 W
1000	62-30	64	N 10-45 W
1200	63-35	65	N 13-15 W
1400	67-48	67	N 17-45 W
1500	67-05	66	N 25-45 W
1700	67-55	65	N 17-00 W
1900	68-00	67	N 9-45 W
2100	69-30	67	N 1-00 E
2300	70-15		
2500	70-00	68	N 16-00 W
2700	69-30	70	N 1-00 E
2900	70-20	70	N 5-30 W
2971	70-10	70	N 9-30 W
3013	68-20	68	N 18-00 W
3051	67-45	68	N 12-00 W
3150	67-45	67	N.G. Glass off
3300	67-10	67	N 39-15 W
3450	66-50	65	N 33-45 W

Table 2 shows the results of the Tro-Pari instrument survey of this hole.

In certain formations where holes tend to deviate, the importance of systematic surveying is obvious. Much valuable drilling may be wasted because the hole fails to reach the desired objective. By careful surveying and wedging it is possible to guide holes that tend to wander. Recent improvements in the art of controlled directional drilling by means of systematic surveying

TABLE 2—*Results of Pitt Gold Mining Company Survey*

Depth, Ft	Dip, Deg	Bearing	Time Setting, Min	Time to Lower Rods, Min
0	85	S 20 E		
350	78	S 35 W	15	9
650	66	S 45 W	20	11
850	64	S 42 W	25	15
950	63	S 45 W	30	19
1200	59	S 46 W	39	20

and wedging have made it possible to explore ore bodies which might never have been found.

The instrument herein described is one of the recent developments available to the mining industry for the improvement in drilling technique.

ACKNOWLEDGMENTS

The writer wishes to acknowledge the courtesy of the Pickands, Mather and Company and Mr. H. W. Johnson for permission to use some of the material for this paper, and the cooperation of E. J. Longyear Company, drilling contractors on this job. Also to Mr. A. C. Lee and Mr. J. H. Evans for their report on the drill-hole survey at Pitt Gold Mining Company's property.

Some Desirable Improvements in Core Barrels

By George D. Roberts*

(New York Meeting, March 1947)

Introduction

Civil engineers are primarily interested in maximum core recovery. This is even more important in foundation work than in mining investigations where sludge samples are of some value. The soft materials, such as clay, gouge, or partially decomposed seams that are most readily ground up or washed away, are the materials that usually have the greatest influence on foundation and leakage conditions. Undisturbed samples of such materials should be obtained for detailed geological examination and laboratory tests. Therefore the attainable goal of sampling in connection with foundation investigations is the recovery of 99.9 pct of undisturbed core.

Factors Causing Core Losses

The principal factors contributing to core losses are: (1) coring badly fractured or very soft material; (2) use of improperly designed coring equipment, including the use of bits of too small a diameter; (3) lack of the proper training and attitude on the part of the drillers and pertinent technical information on the part of the supervising personnel. Little can be done about the character of the material being cored but the other two factors contributing to unsatisfactory performance can be corrected. Although the title of this paper is the improvement and standardization of core barrels, it is believed that the drillers' shortcomings and the lack of pertinent technical literature are more detrimental to good core recovery than either of the other factors.

Uses for Large Diameter Core Barrels

It is well known that the use of larger diameter bits and core barrels generally produce better cores, particularly when sampling soft or broken material. The use of larger and larger bit diameters and the insistence on better core recovery is well demonstrated by the change in practices of the various districts of the Corps of Engineers. Today NX core is the smallest diameter commonly permitted for investigational work and the more active districts are using coring bits up to 6 in. in diameter, as well as 36 and 40-in. calyx rigs. The latter size has proved to be just as economical because it provides more room for core removal and cleanup operations.

The use of large diameter cores to replace the driving and jacking methods of obtaining soil samples is also increasing. The 5⅞-in. Denison sampler, developed by H. L. Johnson to obtain samples from soft formations, is one example. The principal uses for bits larger than NX are: sampling soils and semi-consolidated rock; sampling decomposed rock or rock with soft seams; sampling concrete; obtaining samples for laboratory tests; drilling drainage holes, and other miscellaneous work.

Core Drilling Practices of Several Engineer Districts

Little Rock District

The writer was employed in the Little Rock District from 1938 until 1942. Prior

Manuscript received at the office of the Institute March 17, 1947. Issued as TP 2275 in Mining Technology, November 1947.

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to the subdivision of that District into the Conchas, Tulsa, Denison, and Little Rock Districts, an extremely wide range of materials was encountered in making subsurface investigations at various dam sites. The foundation conditions at the following dams were investigated prior to construction: John Martin (Caddoa), Colorado; Great Salt Plains and Fort Supply, Oklahoma; Denison, Texas; Nimrod, Blue Mountain and Norfolk, Arkansas; and Clearwater, Missouri. In addition to these dams well over a hundred other dam sites were drilled in varying detail.

Originally all the core boring was performed by contract. Early in 1938, however, the District purchased and began operating two Longyear straight-line core drills with hydraulic feeds. The results were so satisfactory that two larger Sullivan rigs, also with hydraulic feeds, were purchased and this was followed by the acquisition of a 40-in. calyx rig. During preliminary investigations, the difficulty of preparing plans and specifications for contract-drilling, which were flexible and fair to both contractor and Government, has produced results not entirely satisfactory and was the primary reason for the change in policy. The policy of the Little Rock District is to perform all of its preliminary investigation drilling and to contract some of the later phases of the subsurface investigations.

This change in policy furnished ample demonstration of a portion of the third reason for core losses, namely, the lack of proper training and attitude on the part of the drillers. In less than a year's time the core recovery at sites, where a recovery of 80 pct was previously considered good, increased to an average overall recovery of about 95 pct. An overall recovery of 99.7 pct was obtained at one site. The percentage of core recovery was computed by dividing the total footage cored into the total footage of core recovered after swell caused by breakage had been subtracted.

The principal rocks cored were moderately hard to hard sedimentary rocks such as limestone, sandstones, shales and so forth. Folding, faulting and solution action had weakened the rock at many of the dam sites. Cementing techniques were developed which eliminated the necessity of reducing to BX core when cavities or caving holes were encountered.

The average overall cost of NX coring by the District was about $5.00 per foot for preliminary investigations. This figure included all applicable costs such as: mobilization, demobilization, diamond loss, amortization of equipment, casing, pressure testing, coreboxes, inspection, supervision, and District overhead. A large part of the success was the result of excellent supervision of the drill foreman, Lewis C. Lindsay, and the change in attitude on the part of the drillers whereby they began to drill for information rather than footage. Strange as it may seem the footage drilled per day increased and 40 ft of core per rig shift was the average on many jobs. Cast-set bortz bits were used exclusively when they proved to be more satisfactory than hand-set carbons. The core barrels were, with two principal exceptions, all double-tube, ball-bearing swivel types that were maintained in excellent condition.

The two principal departures from the conventional core barrels were the use of a Hughes soft-formation barrel and bit and the use of a barrel designed for combination shot and diamond drilling. The Hughes bit was used in the unconsolidated Permian Red Beds in Oklahoma and gave fair core recovery, but some trouble was experienced because of drilling mud entering the barrel along with the core. The combination barrel was purchased primarily to drill the extremely hard vuggy chert, common to the limestone areas, which costs as high $178.00 per foot to core when first encountered. This barrel was definitely unsatisfactory because once the shot was

introduced into the hole it was found impracticable to resume drilling with a diamond bit since some shot always remained in the hole to chew the diamond bit. The completion of the hole by shot drilling was likewise unsatisfactory because of the slow rate of progress. A technique was developed whereby the chert was drilled, at a cost of about $10.00 per foot, by the use of well-worn bits operated at an extremely high speed and pressure. The District is now also using a 6-in. core barrel that is somewhat unsatisfactory because the bit does not cut enough clearance and causes the barrel to bind.

Pacific Coast Districts

The Portland District owns and operates its drilling equipment. Like the Little Rock District the NX cast-set bit and double-tube, ball-bearing swivel core barrels are used almost exclusively. RX bits and roller bits are used to drill in casing seats and reaming shells are commonly employed. Some improvised 6-in. barrels have been used to a small extent. Terrain difficulties serve to emphasize the importance of light weight drilling equipment and in a few instances justifies the use of BX core barrels. The Seattle, Sacramento, San Francisco and Los Angeles Districts contract their core-drilling work although the Sacramento District has recently obtained two drilling rigs.

Use of Core Barrels Larger Than NX

In only a few instances are core barrels larger than NX being used, although there is a definite need for them. The reasons for not using larger barrels are the cost and scarcity of equipment. When good standardized equipment larger than NX is on the market at reasonable prices, and good drilling techniques have been developed, the trend to sizes larger than NX will probably be even more rapid than was the trend from EX, AX, and BX to the now adopted standard of NX. In the case of the Little Rock District all the smaller sizes of core were practically obsolete within a period of two years. The period required for the change throughout the Corps of Engineers was about eight years.

The Concore double-tube core barrel shown in Fig 1 was used with very good success in drilling 3-in. cores of a highly fractured schist that had many soft, decomposed seams. A diamond bit, set with large carbons imbedded in "buttons," was used for cutting. The drill rig was a low-powered 7-hp, hand-feed machine. This same operator was even more successful with this equipment in drilling 4-in. cores of soft, friable and fractured sandstone in connection with maintenance operations at another dam. It is reported that a 6-in. barrel of the same type, and operated by the same rig, recovered an average of 96 pct of the core when drilling a very soft diatomaceous shale deposit for a private company. Some of the holes were several hundred feet deep. The barrel is simple in design and adaptable for use with several types of bits most of which have a bottom discharge. Observation of the drilling operations convinced the writer that, in addition to having a good barrel and rig, an even larger measure of the contractor's success was because of the carefulness, skill and attitude of his drillers.

The Sacramento District personnel have developed a modification of the Denison Sampler. A 6-in. barrel is used with various hard-surfaced cutting bits principally to obtain soil samples. They are operating this barrel with one of the truck-mounted Failing rigs which were developed for the Army Engineer Water Supply Battalions.

The Los Angeles District personnel are experimenting with a 6-in. sampler for cohesionless materials such as sand. The principal innovation is the use of a vacuum to retain the samples.

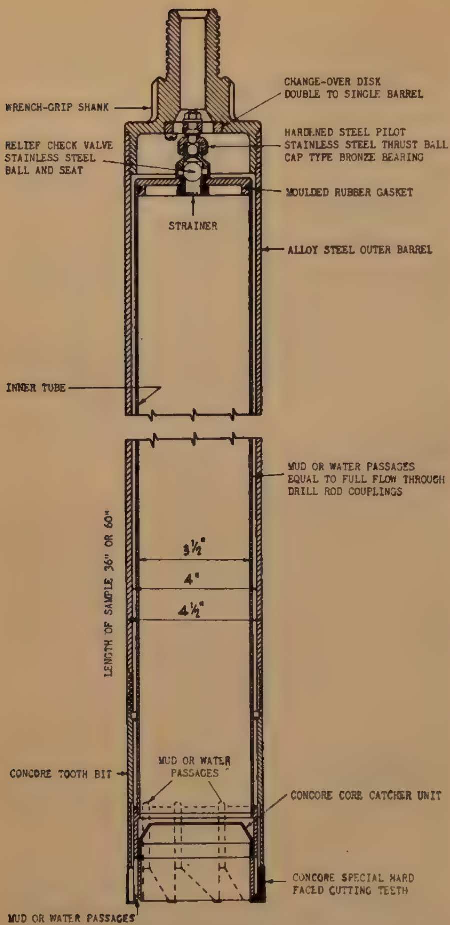

FIG 1—CONCORE DOUBLE-TUBE CORE BARREL. (*Frank L. Howard Engineering Co., Los Angeles, Calif.*)

It will be noted that the inner barrel is not suspended from the head as is usual in core barrels but is only centered there by a pilot and steel ball held in place by the thrust cap. This assembly is also provided with a ball check valve and strainer. The bottom of the inner barrel rides on a brass "wear" ring, which in turn rides on the split-ring base which rides on the shoulder of the bit. This construction is simple and provides three rotating, or slip, surfaces which are readily inspected and cheaply replaced. Inner tubes of different diameters can be used with the outer barrel. The Howard Engineering Co. of Los Angeles manufactures this barrel in the following sizes: EX, AX, NX, 3-in., 4-in. and 6-in. The barrel is used with various types of bits, ranging from hard-surfaced cutting bits to diamond bits.

Résumé of District Practices

Several facts are obvious in reviewing the practices of various districts. First, there is a definite trend away from contract drilling, particularly during the earlier phases of the investigations. Second, there is too large a gap between the now standardized NX core and 36-in. calyx drilling which are the two sizes most commonly employed. Third, there is a definite need for core barrels up to 6-in. in diameter which can be used for either rock drilling or soil sampling.

DESIRABLE FEATURES IN CORE BARRELS

The following features are recommended for incorporation in the designs adopted for standardization.

Double Tube Barrels

It is assumed that the barrels adopted for standardization will be the best type of double-tube, ball-bearing swivel barrels that it is practicable to manufacture and operate. Such barrels should be as simple in design and as rugged in construction as is consistent with obtaining an inner barrel that has very little tendency to rotate, and the avoidance of unnecessarily wide cutting bits. The importance of the latter factor is negligible except in hard-rock drilling.

Bottom Discharge Bits

Since large diameter core barrels are needed to sample soft material it is of paramount importance that the drill water be prevented from washing the sample away. Bottom, or face, discharge bits and a long inner barrel appear to be the best means of protecting the core.

Interchangeable Bits

Large diameter core barrels that are readily adaptable to either hard or soft formation drilling are highly desirable so that the barrel could be used for either

soil, concrete or rock sampling. Interchangeable bits ranging from hard-surfaced, toothed cutting bits to diamond bits would permit this. However, it might be necessary to design a standard outer barrel with two or more standardized and interchangeable inner barrels so that the width of the cutting face on the bit, the type of core holder and the amount and type of drilling fluid could be varied to suit the formation.

Interchangeable Coreholders

Provision should be made in the barrel for the use of several types of coreholders. Improvements on the types of coreholders now on the market would also be of benefit. It seems that a sliding inner barrel could be designed that would actuate the coreholder so that it would not grip the core until the drilling had stopped and the operator was ready to pull.

Indicator of a Blocked Condition

The barrel should be designed so that a blocked bit, or barrel, will be quickly recognized by the driller. The most satisfactory way of providing this indication would be to have the circulation system arranged in such a way that a blocked condition would increase the pressure of the drill water and actuate a blow-off valve. This blow-off valve should be near the rig controls or preferably actuate a switch which would stop the drill's motor.

Prevention of Inner Barrel Rotation

In addition to the ball-bearing swivel head other provisions should be made to prevent, or decrease the tendency of, the inner barrel's rotation. It appears that the flow of drill water between the inner and outer barrels might be utilized to advantage.

Flushing Out Inner Barrel

Drill holes in soft material often have an accumulation of cuttings and so forth at the bottom of the hole, particularly when drilling operations are resumed after a shut down. Provision should be made in the barrel for flushing out such material by water flowing through the inner barrel before coring is resumed.

Removal Calyx

A calyx cup on the barrel would be a decided advantage under certain conditions. Some of the Howard Engineering Company's barrels have a left hand thread at the top of the outer barrel which permits the attachment of a calyx where needed to catch cuttings.

Standardization of Diameters

Bit diameters, thread types and lengths, and the like should be standardized so that parts will be readily interchangeable.

Nomenclature Based on Core Diameter

The designation of core sizes should be based on the core diameters such as 3, 4, and 6 in. The present designation of the smaller diameters such as EX, AX, BX, and NX causes considerable confusion and endless explanations. A numerical designation has been suggested which expresses the diameter of the hole and core in sixteenths of an inch. For example a 122 × 96 bit would cut a hole $7\frac{10}{16}$ in. in diameter and yield a 6-in. core. The adoption of such a nomenclature would be advantageous. The standard casing and pipe sizes should have considerable influence on the size of core barrels adopted for standardization.

Preparation and Distribution of Technical Information

Last but not least, the manufacturers should provide technical data as well as advertising matter. By doing so they would perform a very definite service to their customers. Such technical data would increase the efficiency of coring operations and probably lead to more work as well as more satisfactory specifications for drill-

ing contracts. Harvard University Graduate School of Engineering has made a good beginning toward the gathering of such data.[1] Unfortunately too many of the manufacturers seem to feel that they have complied with a request for information when they drop their latest catalogue into the mail along with some other advertising matter.

Bit Torque

Since most of the core-barrel manufacturers also produce drill rigs the following remarks seem pertinent. It appears that most core-drill rigs are definitely overpowered as far as the torque required on the bit is concerned. The extra power necessary to move skid-mounted rigs, or make deep pulls, is all too often utilized by the driller to grind up core. It therefore appears that shear pins should be provided somewhere in the transmission system. Also a hydraulic transmission system similar to that used by some automobiles might go a long way towards providing a smaller and easily variable torque on the bit. A tachometer at the controls showing the rpm of the bit, and a gauge showing the bit pressure, somewhat similar to the strain gauges used by oil field rotary rigs, would permit closer control of drilling and consequently improve core recovery.

Engineering the Drilling

As is well known no two drilling jobs should be considered as identical. Each foundation, for example, has certain minor peculiarities which make the drilling conditions different. To obtain maximum core recovery the drillers must experiment until they find the most efficient combination of drilling techniques, bits, bit clearances, modifications of core holders and the like, that will recover the most core on that particular job. The writer has found it practically impossible to write drilling specifications that will cover this factor and yet be fair to both the contractor and the Government. A sort of irreconcilable conflict appears to exist between drilling for information and drilling for footage. Footage drilled without good core recovery and drilling data is footage and money wasted. All too many people fail to recognize that both core recovery and footage can be obtained and that when this is done operating costs usually drop—as in the case of the Little Rock District.

Bore-hole Camera

The Corps of Engineers is interested in the development of a bore-hole camera which may have considerable influence on core drilling. The feasibility of the camera and projector have been demonstrated by working models. However, the company that was developing the camera, under an agreement with the Chief of Engineers, is now too interested in oil field equipment to continue experiments which would lead to the commercial production of such cameras and projectors. They have agreed to relinquish their patent rights to anyone producing the camera provided that they are allowed to purchase a number of the cameras and projectors for their own use at the cost of manufacture. Anyone interested in the manufacture of this equipment should communicate with Mr. E. B. Burwell, Office, Chief of Engineers, U. S. Army, Civil Works Branch, Gravelly, Virginia.

The principals of the camera are shown in Fig 2. A movie camera takes pictures of the walls of the core hole as reflected by the mirror. Depth and azimuth indicators identify and orient each picture. The developed film is projected by reflection from a conical mirror onto a ground glass cylinder. Thus, the pictures showing the inside of the core hole

[1] The Present Status of the Art of Obtaining Undisturbed Samples of Soils. Pub. No. 281, Harvard Univ. Grad. School of Eng.

appear on the outside of a cylinder of the same diameter. It is hoped to develop this camera for use in NX core holes, however, the use of larger diameter holes would be more convenient as well as more expensive.

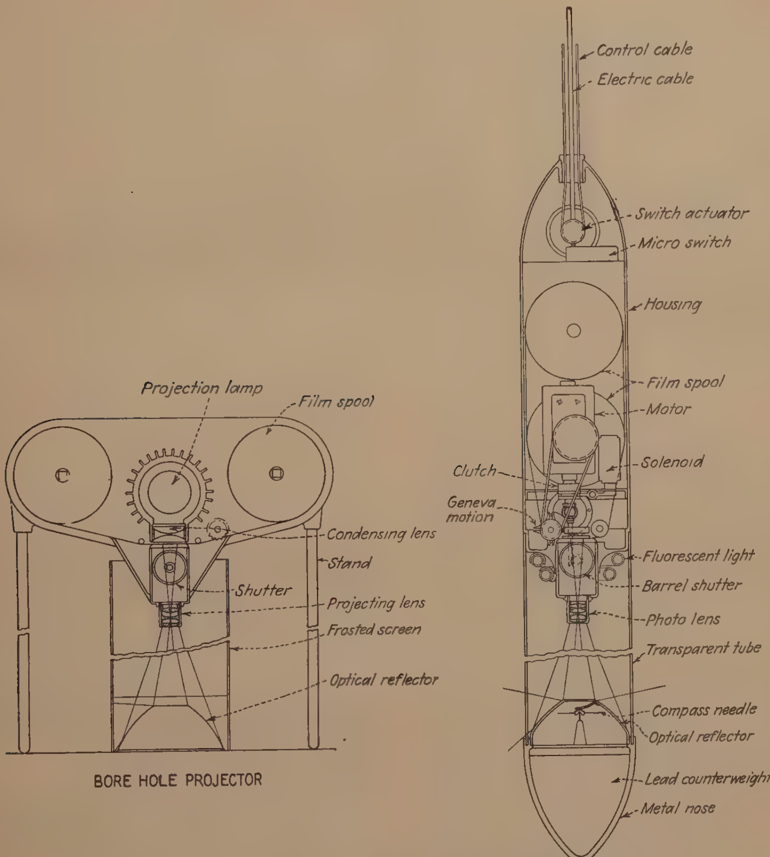

FIG 2—PRINCIPALS OF BORE-HOLE CAMERA.

ACKNOWLEDGMENTS

The following personnel of the Corps of Engineers provided suggestions for incorporation in this paper: Messrs. A. S. Cary and R. L. Nichols, Seattle District; L. L. Ruff and C. J. Monahan, Portland District; V. P. Pentegoff and M. K. Reade, Los Angeles District; V. L. Glaze, Sacramento District; and Harold Stuart of the Little Rock District. Mr. Frank L. Howard of the Howard Engineering Co., Los Angeles, also provided information on that Company's Concore barrel and drilling procedures.

Practice of Omaha District, Corps of Engineers, War Department, in Recovering Cores between Two and Ten Inches in Diameter

By John H. Melvin,* Member AIME

(New York Meeting, March 1947)

The Omaha District, Corps of Engineers, has been doing subsurface exploration work for a number of years, both by contract and with its own forces. Certain practices and procedures concerning the recovery of large diameter cores have been developed which will be described in this paper.

Geology

Most of the drilling to date has been in connection with structures on the main stem of the Missouri River above Sioux City, Iowa. A wide variety of subsurface conditions has been encountered. The bedrock formations appear in regular order from the oldest to the youngest, traveling upstream; or in other words, the general regional dip is to the northwest.

The oldest formation is the Carlile shale, a compaction or noncemented waxy shale of Cretaceous age. Because of its uncemented, soft nature this shale has a tendency to grind or wash away when drilled. It also has a tendency to rebound or expand when the overlying pressures are removed.

A 25-ft bed of sharp, friable sandstone called the Codell member is present near the top of the Carlile shale. This member is a cemented sediment, but in drilling, many of the sand grains are broken loose and exert extensive abrasive action on the core barrel and bit.

Manuscript received at the office of the Institute March 31, 1947. Issued as TP 2295 in Mining Technology, March 1948.

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Above the Carlile shale is the Niobrara chalk, a soft calcareous deposit with shaly horizons and some thin beds of bentonitic clay. The average thickness of this deposit is about 145 ft. It is soft enough to be sawed or bored with ordinary woodworking tools, yet is compact enough to be used as building block.

The Pierre shale, the formation above the Niobrara chalk, is over 800 ft in thickness and is the most widespread material in the area. It is a dark colored, compaction shale containing numerous thin bentonitic clay seams and at certain horizons, hard manganese concretions. The shale has numerous faults and in addition, along the valley walls, has undergone considerable slumping. Joints, fault planes and slickensided surfaces are common features encountered in drilling and add to the difficulties of high core recovery.

East of the Missouri River the bedrock is generally covered with from 100 to 200 ft of glacial till, composed of clays, silts, sands, gravels and boulders. The valley itself is filled with alluvial deposits, mostly of the sand and gravel sizes, and contains a buried preglacial or interglacial channel over 100 ft in depth. The overburden deposits require sampling in connection with engineering studies and must also be cased before the underlying bedrock can be cored. The diameter of the required overburden samples can thus have an indirect effect on the size of core recovered from the bedrock. West of the river, which in most cases is the western boundary of glaciation,

the bedrock is generally not covered with overburden, although in some localities wind blown loess or unconsolidated Tertiary deposits require coring.

Development of Core Barrel of Larger Diameters

In retrospect, our present practice has been a gradual development rather than a sudden change from one type of equipment to another. *NX* double-tube core barrels gave very poor core recovery in the relatively soft rocks of the Missouri River Valley. A 3-in. single-tube core barrel was tried briefly but gave little improvement. Next, a single tube barrel some 7 ft in length was made from ordinary 6-in. casing. This barrel gave very satisfactory recovery in the Niobrara chalk but was of little use in the shale formations.

Finally in 1940, a double-tube core barrel somewhat similar to the Dennison-type sampler was designed. It cut a $5\frac{15}{16}$-in. core and had an outside diameter of $7\frac{5}{8}$ in. It was in general constructed along the lines of an ordinary double-tube core barrel and was designed to cut 5 ft at each run. Standard core drilling equipment and *N* rods were used. The bottom discharge bit was set with Haystellite slugs because ordinary diamond bits had a tendency to "gum up" and also because the slugs were more economical. The rebound or expansion characteristics of the shales ordinarily cause some caving, particularly in the larger holes, so drilling mud was used instead of water with this barrel.

At first an ordinary ring type core lifter was used. Some core was lost because of the large size and also because the sharp sand from some horizons eroded a groove in the taper bit which prevented the lifter from performing in the ordinary manner. A special lifter was devised consisting of the regular ring lifter to which was attached 8 steel strips $\frac{1}{2}$-in. wide and rounded at the top. This really formed a combination ring and basket type lifter and has been very successful. This particular type of lifter has now been patented.

The double tube barrel could also be adapted for the sampling of various types of soils. An ordinary stove pipe sample tube was placed within the inner barrel and was held in place by a $\frac{1}{8}$-in. flange on the inner bit. Extensions of various lengths were provided for the inner barrel so that a cutting or drive shoe extended beyond the rotating outer bit. As the outer bit is rotated, the inner bit is jacked into the formation being sampled. The type of cutting teeth on the rotating bit and the length of the inner bit depends on the characteristics of the materials being sampled.

It was intended that the described core barrel would be used through standard 8-in. pipe. Since many holes penetrate 100 to 200 ft of overburden before reaching bedrock, there are times when the casing is not entirely straight. This is particularly true if short lengths have been used. In addition, drilling mud assures a tighter fit. If double strength pipe is used, the inside clearance is cut down even more.

The above casing requirements caused a reduction in the dimensions when later barrels were built. The outside diameter was reduced from $7\frac{5}{8}$ to 7 in. The core size was reduced from $5\frac{15}{16}$ to $5\frac{3}{8}$ in. Other features have been retained.

We are heartily in favor of standardization of core barrels of the larger diameters. In our work in the Missouri River area, a barrel recovering a core in the vicinity of 6 in. in diameter would be the most desirable. In order to utilize 8-in. heavy duty pipe for casing the holes, our experience indicates that the maximum outside diameter of the cutting teeth on the bit should not be over $7\frac{1}{4}$ in.

If a barrel can be designed which will take a 6-in. core and still not have an outside diameter greater than $7\frac{1}{4}$ in., our needs will be taken care of. Otherwise, we

would favor a slightly smaller core so as to retain an outside diameter which would be efficient under our actual field conditions.

ACKNOWLEDGMENT

Thanks are due Brigadier General Lewis A. Pick, co-author of the Pick-Sloan plan for the comprehensive development of the Missouri River Basin, now Division Engineer, Missouri River Division and Lt. Col. Delbert B. Freeman, District Engineer in charge of the Omaha District where the above described subsurface exploration technique was developed.

The Resolving Power of Magnetic Observations

By Irwin Roman*

(Chicago Meeting, February 1946)

In studying the possibilities of a continuously recording magnetometer for use along the surface of the earth and in an airplane, the Federal Bureau of Mines was led to a study of the theoretical resolving power of magnetic observations, both in the variation of the total magnetic intensity and in each of the two components ordinarily measured. Along with this general study, an attempt was made to determine the relative worth of the total intensity anomaly and the vertical anomaly in geophysical prospecting. General considerations led to three conclusions:

1. In the northern magnetic hemisphere of the earth, the total intensity anomaly is offset to the south of the vertical anomaly, for a positive anomaly (see Appendix, note 1).

2. The resolving power of magnetic observations decreases rapidly with distance from the disturbing body.

3. The horizontal anomaly has little value in prediction of the disturbing body but might be useful in making a choice between bodies suggested by the vertical or total anomaly.

Some of the considerations leading to these conclusions were:

1. The normal intensity and the anomalous intensity are vectorially additive; hence for a positive anomaly in the northern magnetic hemisphere the total intensity will be a maximum at some point south of the disturbing body and north of the point at which the line to the disturbing body is parallel to the normal earth's field. The anomaly itself has a maximum value directly above the body. For a positive anomaly of constant magnitude, the total intensity will be a maximum when the anomaly vector is parallel to the normal vector. This is shown schematically in Fig. 1 (not to scale) for a magnetic distribution not restricted to a relatively thin sheet in the magnetic meridian. The normal field has a magnitude I_0 with vertical component V_0 and horizontal component H_0. Six station points along a traverse across the body from south to north are shown at M to S. At each of these stations the normal magnetic field vector terminates at F. At each of these stations the magnetic field vector due to the disturbing body B terminates at A. The line through the station point and the point A passes through B. The magnitude of the anomaly, indicated by the length of the vector terminating at A, is proportional to some power of the distance from the station point to the disturbing body. If the disturbing body B is a concentrated point of magnetic material, the anomaly is proportional to the inverse second power of the distance. For a uniform magnetic distribution over a line perpendicular to the observed traverse, the anomaly is proportional to the inverse first power of the distance. For surface distributions, the law of intensity is more complicated. At each field point, the total magnetic intensity T is the vector sum of the

Published by permission of the Director, Bureau of Mines, U. S. Department of the Interior. Manuscript received at the office of the Institute Feb. 9, 1946. Issued as TP 2097 in Mining Technology, November 1946.

* Senior Geophysicist, Bureau of Mines, Baltimore, Maryland.

normal field and the anomalous field, and its magnitude depends on the directions as well as on the magnitudes of the two fields. If P is the station for which PB is increasing so that the magnitude of T and of the anomaly is increasing as the station passes through P from south to north. At Q, the magnitude of A is a

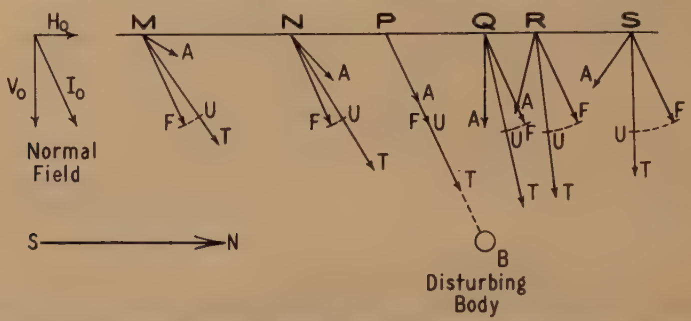

FIG. 1.—SCHEMATIC ANALYSIS OF TOTAL ANOMALY FOR A POSITIVE DISTURBANCE IN THE NORTHERN MAGNETIC HEMISPHERE (NOT TO SCALE).

is parallel to the normal earth's field, the vector A will be to the south or north of the vector F, according as the station is north or south of P. The total intensity vector will lie between A and F. If Q is directly over B, the magnitude of A will be a maximum at Q and will fade as the station recedes from Q in either direction. The angle between A and F is zero at P and increases as the station recedes from P in either direction. If U is a vector parallel to T with a magnitude equal to that of F, the length from U to T represents the anomaly in the total intensity. As the station approaches P from the south, the magnitude of A increases and the angle between F and A decreases, so that the vector T increases in magnitude. As the magnitude of U is constant, the anomaly UT increases as the magnitude of T increases. As the station recedes from Q to the north, the magnitude of A decreases and the angle between A and F increases so that the total intensity and its anomaly decrease. As the station moves from P to Q, the magnitude of the anomaly A and that of the angle between F and A both increase. Near P, the cosine factor is nearly constant and the magnitude of A maximum, so that it changes only slightly as the station passes through Q. However, the angle between F and A is increasing so that the total intensity is decreasing as the station passes through Q from south to north. At some point between P and Q, the rate of increase in the magnitude of A just balances the rate of decrease in the cosine of the angle between F and A, so that the total intensity vector T has a maximum value. The exact value and position of this maximum depend on the normal field vector, and on the relation expressing the law of the anomaly. However, the maximum value in the total intensity anomaly always occurs between P and Q. The trivial case in which P and Q coincide is no exception.

2. As the distance of the disturbing body below the traverse of observation increases, the anomaly curve becomes broader and flatter. For multiple sources of the anomaly, the total intensity is the vector sum of the normal field and each separate anomalous field. The total anomaly is the sum of separate effects, each of which adds to the normal field a quantity that changes more slowly along the traverse as the depth of the body increases. Hence, the resolving power,

or the ability to separate the effects of different disturbing bodies, decreases as the depth increases.

3. As the horizontal anomaly for the assumed conditions is northward to the south of the body and southward to the north of the body, it results in partial counterbalancing of effects from different bodies. This results in smaller anomalies between the bodies and increases the effects of errors of observation. The horizontal anomaly has a positive maximum to the south of the body, is zero over the body, and has a negative minimum to the north of the body. These offsets result in loss of identifiable characteristics for multiple disturbing bodies.

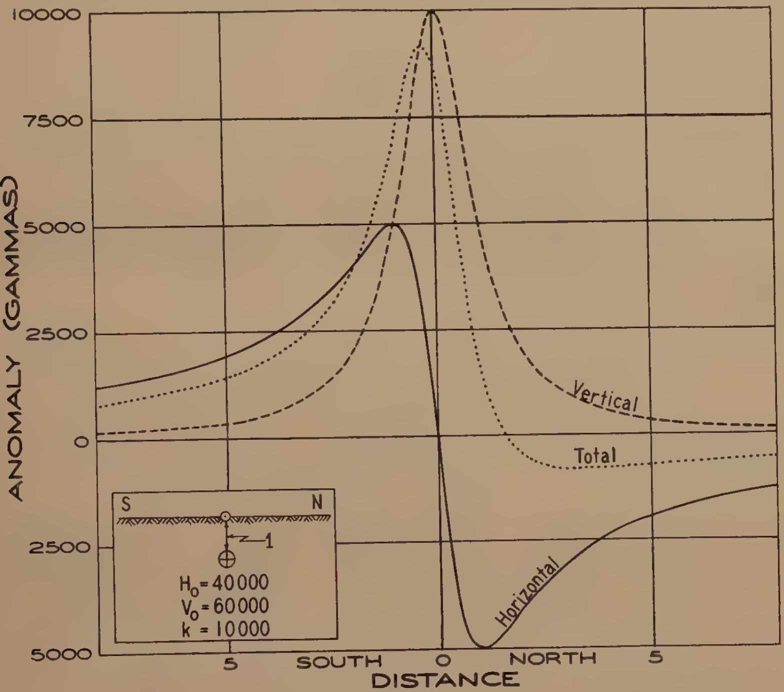

Fig. 2*a*

FIGS. 2*a* to 2*d*.—ANOMALY CURVES FOR A HORIZONTAL LINE POLE LYING EAST-WEST AT UNIT DEPTH ALONG A SOUTH-NORTH PROFILE.

TYPICAL BODIES

To illustrate quantitatively the conclusions reached qualitatively in the foregoing paragraphs, certain typical cases were analyzed numerically. The variables involved include the normal field, the strength-law of the anomaly, the location of the body, and the locations of the observation stations. Because of the multiplicity of possible combinations, only a few cases could be considered (see Appendix, note 2). The normal field was taken as being of an order roughly applicable over the United States. In three simple cases, this was replaced by values representing roughly the maximum horizontal and the maximum vertical intensity over most of the country. The disturbing body was considered as made up of one or more horizontal line poles lying east-west, each uniformly magnetized over its entire length. The line of traverse was taken as north-south along the surface of the earth. These assumptions simplified

the calculations in various ways. Geologically, the bodies may be taken as relatively thin dikes striking magnetically east-west and dipping parallel to the normal field. The effects of the sides are, therefore, negligible. The lower edges of the dikes are considered deep enough to be neglected.

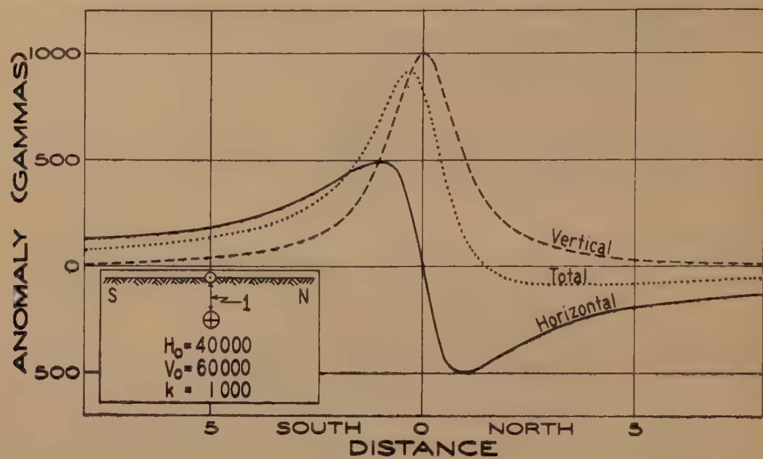

Fig. 2b

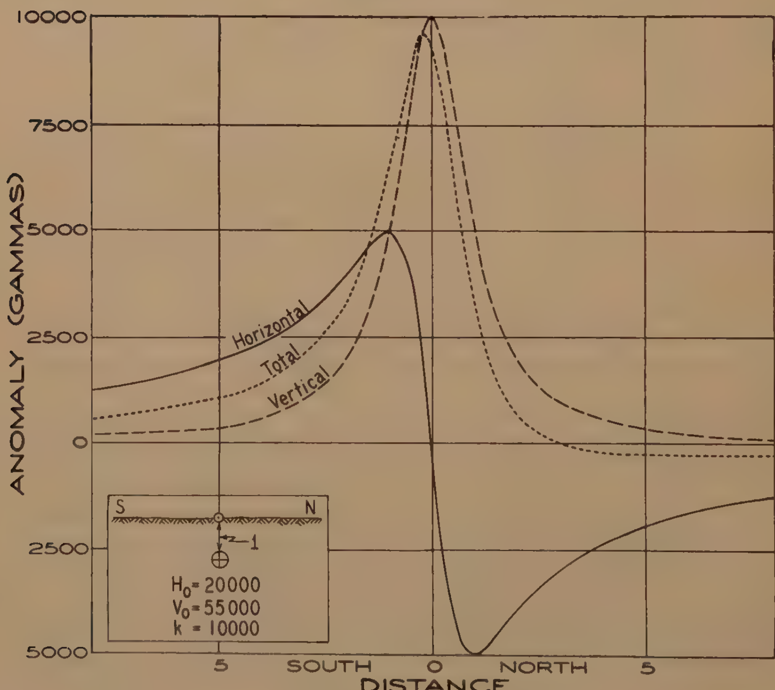

Fig. 2c

The conclusions reached from these idealizing assumptions have been verified by more elaborate calculations for distributed magnetic bodies and also by field

experience. The agreement between the three has been sufficiently close to justify the conclusions reached, for a positive anomaly in the northern magnetic hemisphere.

For the idealized conditions, we have:

H_0 = the normal horizontal magnetic intensity.

V_0 = the normal vertical magnetic intensity.

x = the distance of the field station north of the line directly over the body.

z = the depth of the body below the surface of observation.

k = the effective pole strength of the body, defined numerically as the attraction due to the body at a unit distance from the body and dimensionally as the product of a magnetic force and a unit distance.

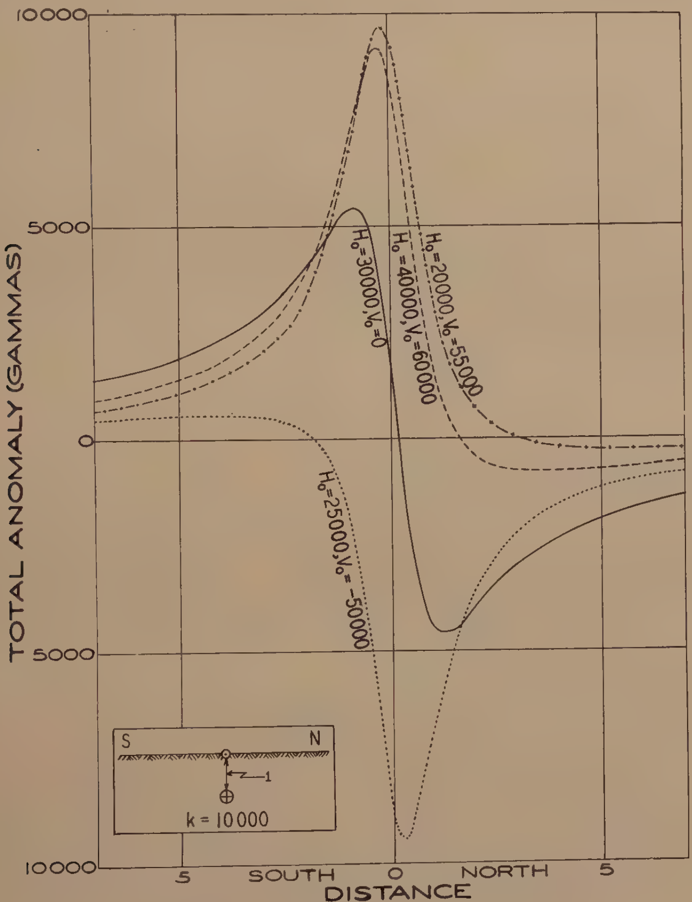

Fig. 2*d*

For these conditions and notations, the anomaly has the components:

$$\text{Horizontal: } H_a = -\frac{kx}{x^2 + z^2}$$

$$\text{Vertical: } V_a = \frac{kz}{x^2 + z^2}$$

and the total field has the components:

$$\text{Horizontal: } H = H_0 + H_a$$
$$\text{Vertical: } V = V_0 + V_a$$

The anomaly has a value

$$I = \frac{k}{\sqrt{x^2 + z^2}}.$$

The total intensity is

$$T = \sqrt{H^2 + V^2}$$

and the "total anomaly," defined as the excess of the magnitude of total intensity over the magnitude of the normal field, is

$$T_a = T - T_o = \sqrt{H^2 + V^2} - \sqrt{H_o^2 + V_o^2}$$

Theoretically it is possible to transform the expression for this total anomaly into more useful forms for specific purposes, but for purposes of numerical computations the direct form was used.

For definiteness, all intensities will be taken in gammas and all lengths will be taken in an arbitrary unit. Thus the value to be assigned to k for an actual body would depend on the unit of length, but this should cause no confusion.

For a preliminary analysis, the normal field was taken as $H_0 = 40{,}000$, $V_0 = 60{,}000$, in gammas, and the depth was taken as one unit. Fig. 2*a* shows the variations of the vertical, horizontal and total anomalies as the point of observation moves across the position of the pole, for an effective pole strength of 10,000. Fig. 2*b* shows the variations for an effective pole strength of 1000 with the same normal field. Fig. 2*c* shows the variations for an effective pole strength of 10,000 in a normal field $H_0 = 20{,}000$, $V_0 = 55{,}000$. Inspection of these three figures shows a remarkable similarity. In fact, if the three sets of curves are drawn to the same scales and the values computed for $k = 1000$ are multiplied by 10, each of the component curves of each figure coincides with the corresponding curves on the other two figures while the total curves have ordinates roughly inversely proportional to the magnitude of the normal field. This is evident for the vertical and horizontal components as the values of H_a and V_a for a selected pair of values of x and z are each proportional to k and independent of H_0 and V_0. For these two components the agreement is theoretical. For the total anomaly, the agreement is only approximate. However, the differences will be so small as to be negligible for practical purposes and not detectable in ordinary graphs, except away from the body.

The total anomaly is:

$$\begin{aligned} T_a = T - T_0 &= \sqrt{H^2 + V^2} - \sqrt{H_0^2 + V_0^2} \\ &= \sqrt{H_0^2 + V_0^2}\left\{\sqrt{\frac{H^2 + V^2}{H_0^2 + V_0^2}} - 1\right\} \\ &= T_0\{\sqrt{1 + \varphi} - 1\} \end{aligned}$$

$$\begin{aligned} \text{where } \varphi &= \frac{2H_0H_a + 2V_0V_a + H_a^2 + V_a^2}{H_0^2 + V_0^2} \\ &= \frac{k}{(x^2 + z^2)(H_0^2 + V_0^2)}\{2V_0z - 2H_0x + k\} \end{aligned}$$

$$\begin{aligned} T_a &= T_0(\tfrac{1}{2}\varphi - \tfrac{1}{8}\varphi^2 + \cdots) \\ &= \tfrac{1}{2}T_0\varphi(1 - \tfrac{1}{4}\varphi + \cdots) \end{aligned}$$

If k is small, φ is small and almost proportional to k if the other variables are fixed so that T_a is almost proportional to k for each selected set of values (x,z,H_0,V_0). Since φ is roughly inversely proportional to T_0^2, T_a is roughly inversely proportional to T_0 for each selected set of values (k,x,z), although not accurately so.

Figs. 2*a*, *b*, *c* show that the total intensity has an anomaly with a maximum to the south of the line of poles and that the vertical component has an anomaly with its maximum directly over the line of poles. The magnitude of the maximum total anomaly is slightly less than that of the vertical component. Also, the total anomaly is negative as the station recedes

to the north and has a negative minimum at some point north of the line of poles. To investigate the "offset" of the maximum total anomaly south of the maximum vertical component, consider the rate at which the total intensity increases as x increases. The total intensity is:

$$T = \sqrt{H^2 + V^2}$$
$$= \sqrt{H_a^2 + V_a^2 + 2H_0H_a + 2V_0V_a + H_0^2 + V_0^2}$$

For the line pole selected,

$$H_a = -\frac{kx}{r^2} \quad \text{and} \quad V_a = \frac{kz}{r^2}$$
$$\text{where } r^2 = x^2 + z^2$$

Hence:

$$\frac{\partial H_a}{\partial x} = \frac{k}{r^4}(x^2 - z^2) \quad \text{and} \quad \frac{\partial V_a}{\partial x} = -\frac{2kxz}{r^4}$$

Accordingly:

$$\frac{\partial T}{\partial x} = \frac{1}{2T}\left(2H_a\frac{\partial H_a}{\partial x} + 2V_a\frac{\partial V_a}{\partial x} + 2H_0\frac{\partial H_a}{\partial x} + 2V_0\frac{\partial V_a}{\partial x}\right)$$
$$= \frac{kH_0}{r^6T}(x^2 + z^2)(x^2 - z^2 - \lambda x)$$
$$\text{where} \quad \lambda = \frac{k + 2V_0z}{H_0}$$

Thus T has extrema for

$$x = \tfrac{1}{2}(\lambda \pm \sqrt{\lambda^2 + 4z^2})$$

If we write $x_1 = \frac{1}{2}(\lambda - \sqrt{\lambda^2 + 4z^2})$

$$x_2 = \tfrac{1}{2}(\lambda + \sqrt{\lambda^2 + 4z^2})$$

then $\dfrac{\partial T}{\partial x}$ is positive outside the range (x_1, x_2), is negative on that range and is zero at the extremities of that range. Thus T has a maximum at $x = x_1$ and a minimum at $x = x_2$, for $k > 0$.

For $k < 0$, T has a minimum at $x = x_1$ and a maximum at $x = x_2$. This analysis assumes that $H_0 > 0$.

The positions of the extrema are shown for the selected normal fields in Figs. 3*a*, *b*. The abscissa in each graph represents the effective pole strength. The curve rising to the right shows the abscissa of the minimum to the north of the line of poles and the corresponding scale is shown at the right edge of the graph. The curve dropping to the right shows the abscissa of the maximum to the south of the line of poles and the corresponding scale is shown at the left edge of the graph. As the curve is relatively flat at its minimum, as the ordinate of this minimum is small compared with the ordinate of the maximum and as the minimum would be absorbed in the maxima of near-by poles, the principal interest lies in the position of the maximum, which is relatively sharp. For $H_0 = 40{,}000$ and $V_0 = 60{,}000$, the maximum is offset to the south by a distance varying from about 0.3 unit for small values of k to about 0.2 unit for an anomaly approximately equal to the normal total intensity. For $H_0 = 20{,}000$ and $V_0 = 55{,}000$, representing more typical conditions in continental United States, the offset varies from less than 0.2 to about 0.1 unit. Thus, the offset of the total anomaly is to the south of the vertical anomaly by an amount not much in excess of three tenths of the depth and probably considerably less.

As examples of cases excepted from the previous discussion, Fig. 2*d* shows the variation of the total anomaly for four cases, the two shown in Figs. 2*a* and 2*c*, one at the magnetic equator, the fourth in the southern magnetic hemisphere. As the vertical and horizontal anomalies are the same as those shown in Figs. 2*a* and 2*c*, these curves have been omitted. For the position at the magnetic equator, the curve follows the horizontal anomaly in its trend, although the two are not identical. For the position in the southern magnetic hemisphere, the curve is almost symmetric in the origin to the corresponding curve in the northern magnetic hemisphere. Hence the positive maximum would be replaced by a negative minimum.

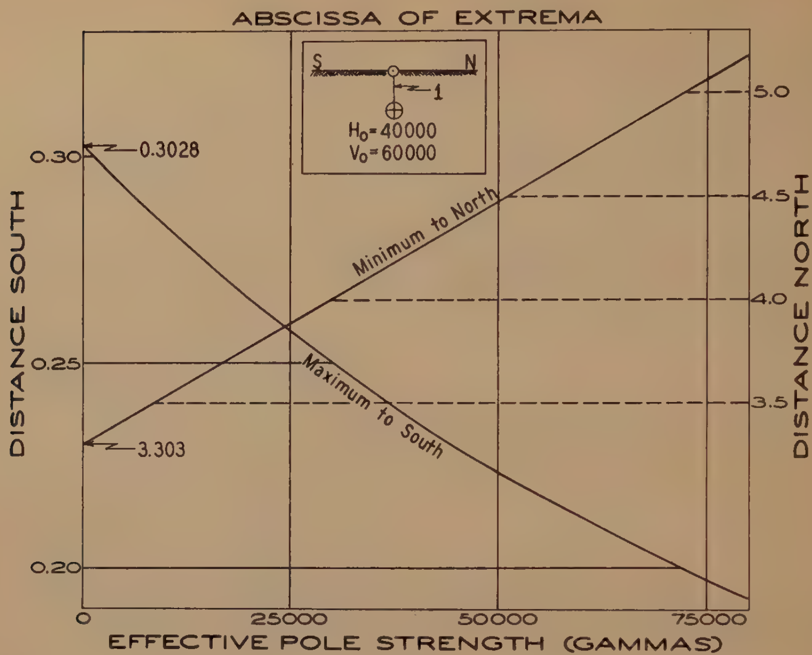

Fig. 3a

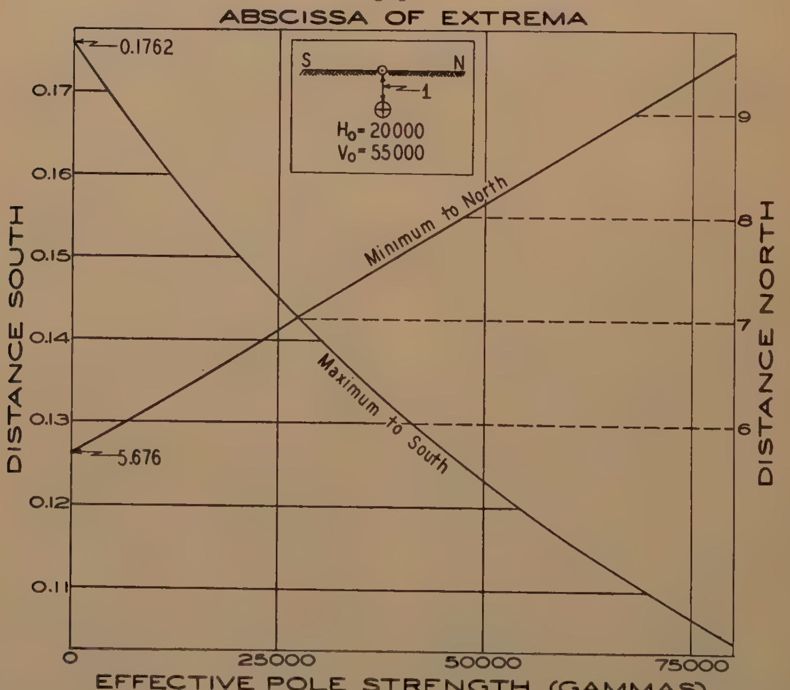

Fig. 3b

FIGS. 3a and 3b.—VARIATION OF ABSCISSA OF EXTREMA AS POLE STRENGTH OF DISTURBING BODY VARIES.

Several typical cases of the cumulative effects of multiple line poles were studied. Fig. 4 shows the anomaly curves over a pair of line poles with a positive pole to the south and a negative pole to the north. The vertical component is positive to the south of the mid-point, negative to the north and zero over the mid-point. The horizontal anomaly is positive at both ends and negative between two points $x = \pm \frac{1}{2}\sqrt{5}$. The total anomaly is offset slightly to the south, its maximum being somewhat smaller than the vertical intensity maximum and its minimum numerically somewhat larger than the minimum of the vertical intensity.

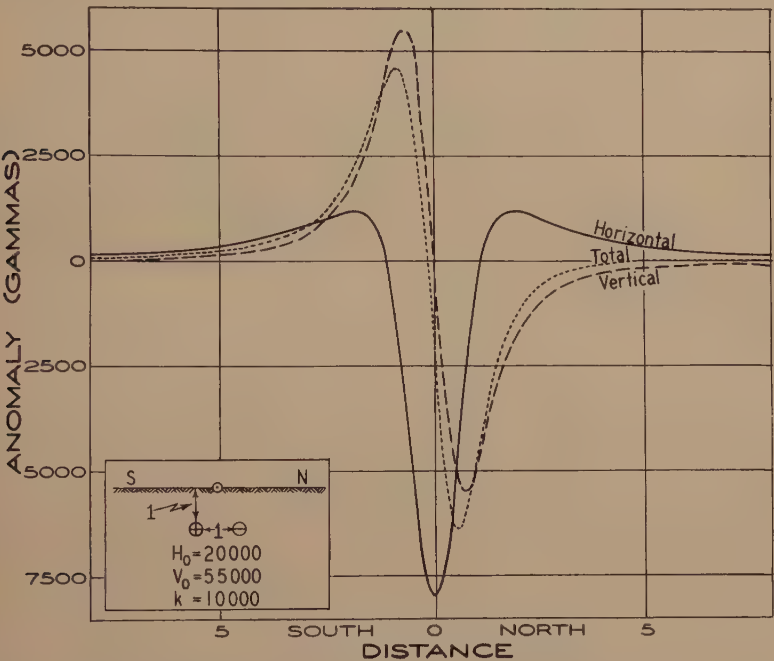

FIG. 4.—MAGNETIC ANOMALIES OVER A LINE DIPOLE ONE UNIT DEEP AND ONE UNIT LONG.

Figs. 5*a* to *e* show the effects of a pair of positive line poles buried at various depths. In each figure the poles are located at $x = -2$ and $x = +2$. Fig. 5*a* shows the effects for a depth $z = 1$. The vertical component has a maximum over each pole, with the same numerical value. The total intensity has a maximum slightly offset to the south of each maximum of the vertical component. The more southerly of this pair has a larger value than the vertical anomaly, and the more northerly has a value slightly smaller than the vertical anomaly numerically. The horizontal intensity shows the typical interaction that makes it difficult for purposes of identification but useful for verification of structures assumed on the basis of the vertical or total anomaly. It has two maxima and two minima.

Fig. 5*b* shows the effects for a depth $z = 2$. The two equal maxima of vertical anomaly occur over the two poles but are about half as large, with the intervening minimum much closer to the value of the two maxima. The identity of the two poles is thus much less distinct. The total anomaly is offset as before and consists of a pair of peaks, the southerly one being higher and the northerly one lower

than the vertical peaks. The intervening minimum is lower than that of the vertical curve. The horizontal curve blends the two anomalies still more and consists of a single southerly maximum and a northerly minimum.

Fig. 5*c* shows the effects for a depth $z = 3$. The minimum between the two maxima of vertical intensity has almost disappeared, the pair of poles being recognizable only by a broadened peak. The total anomaly is also broadened and its two peaks are barely discernible, the top being a flattened slope. The horizontal

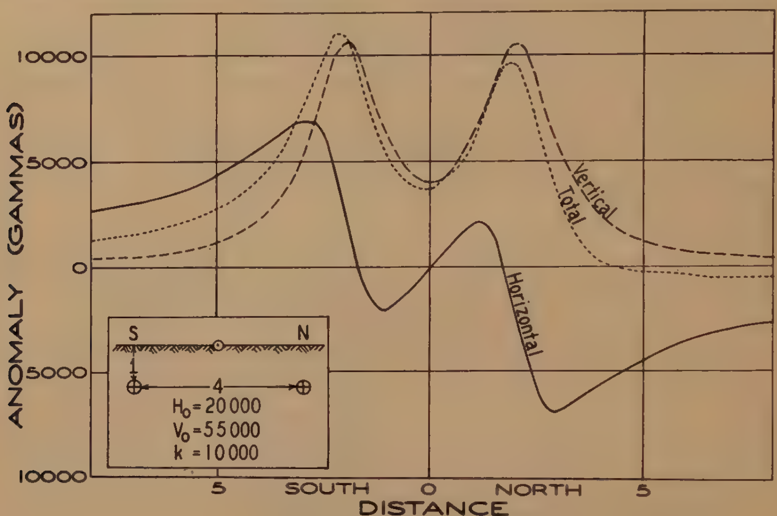

Fig. 5*a*

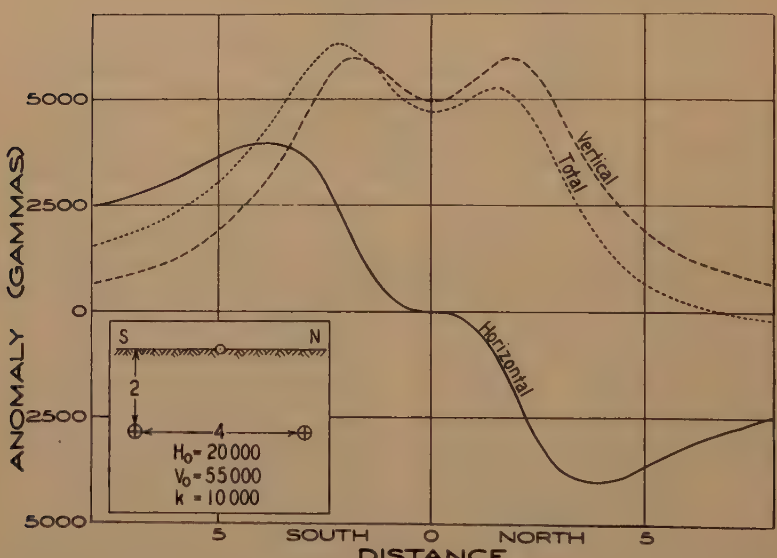

Fig. 5*b*

FIGS. 5*a* to 5*e*.—MAGNETIC ANOMALIES OVER A PAIR OF POSITIVE LINE POLES FOUR UNITS APART AT VARIOUS DEPTHS.

anomaly curve is also almost that of a single pole but much broader in appearance.

Fig. 5*d* shows the effects for a depth

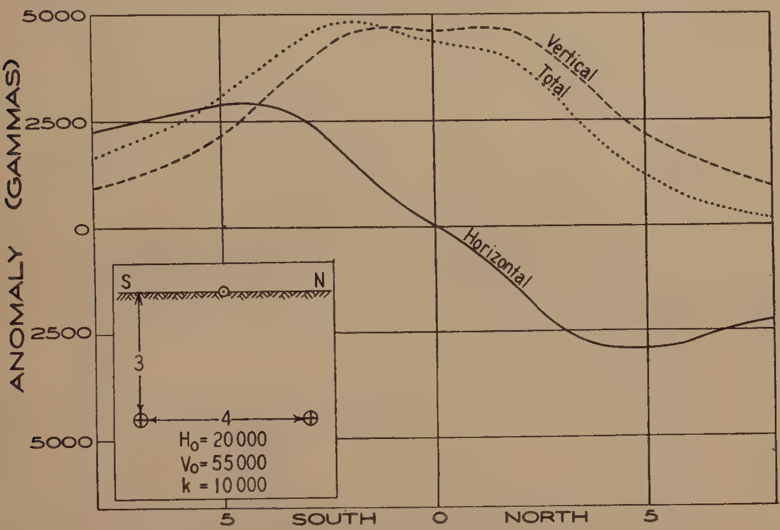

Fig. 5*c*

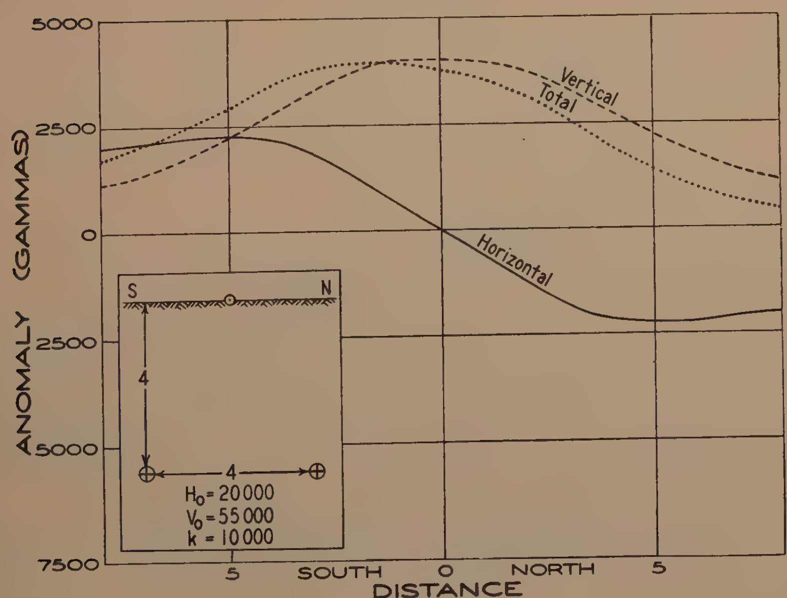

Fig. 5*d*

$z = 4$. Except for horizontal broadening, each of the three curves resembles that for a single pole. However, any drilling based on this conclusion might be costly, as such a curve must be analyzed more carefully, since a drill hole over the "magnetic high" would fail to find either of the two "ore bodies."

Fig. 5*e* shows the effects for a depth of

$z = 5$. The lack of resolving power is still more marked. The magnitude of the effects also becomes smaller as the depth increases.

To further accentuate the loss of resolving power as the depth increases, the vertical anomaly curves are collected in Fig. 6*a* and the total anomaly curves in Fig. 6*b*. Relative magnitudes are clearly shown along with changes in characteristics. As the horizontal component has questionable identification value, no composite figure has been prepared for it.

In Figs. 7*a* and 7*b*, actual survey shows the comparison between the total intensity and the vertical intensity. Two magnetic highs have been selected for comparison. In anomaly I, the offset is about ¾ miles almost due south. In anomaly II, the offset is about 2¼ miles, approximately south-southeast. The first offset can be explained within observational errors. The second offset is so large as to require further investigation before agreement can be reached. Although no wells have been drilled on either high, depths are indicated as about a mile with considerable possible variation.

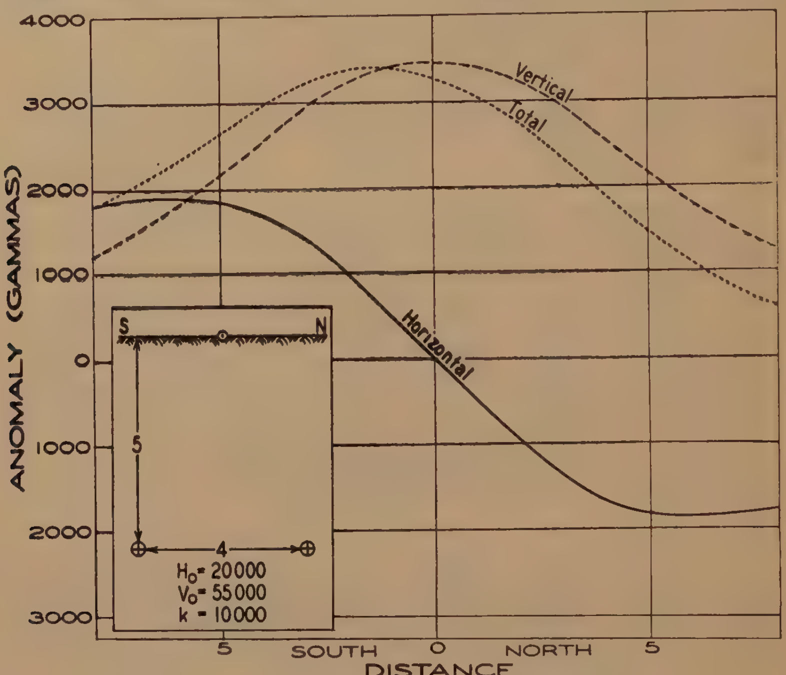

Fig. 5*e*

Conclusions

The qualitative conclusions have been verified by theoretical calculations and agree with general experience over a number of years. They are:

1. In the magnetically northern hemisphere, the total anomaly for a positive disturbing body is offset to the south of the body.
2. The resolving power of magnetic observations decreases rapidly with the depth of the body.
3. The horizontal anomaly may be used for verification of predictions but has questionable identification value.
4. In the magnetically southern hemisphere, the normally induced magnetiza-

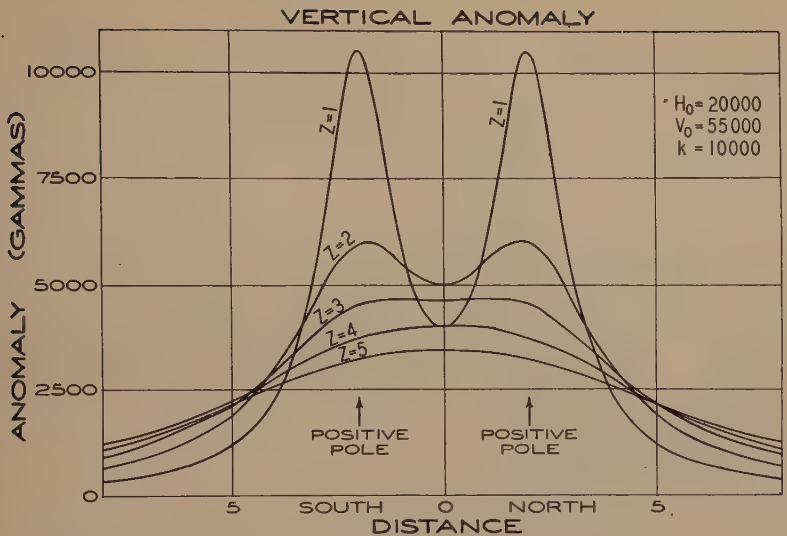

Fig. 6*a*

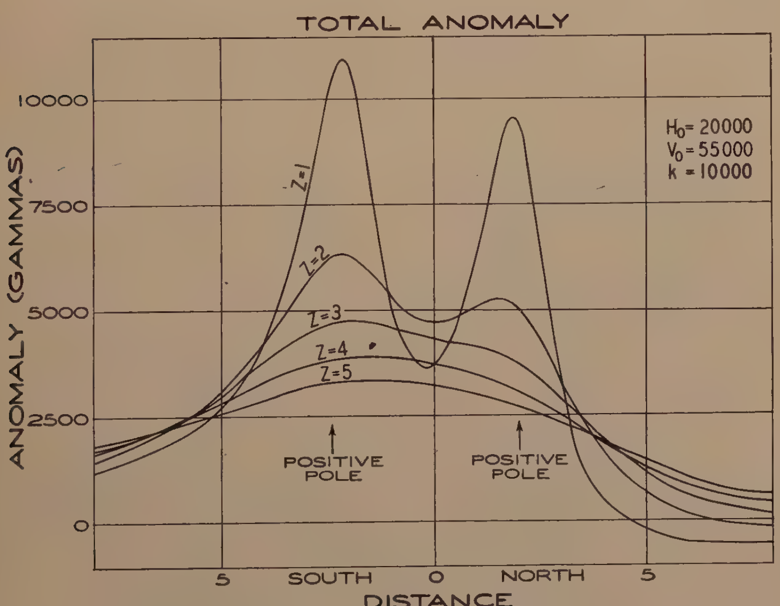

Fig. 6*b*

FIGS. 6*a* and 6*b*.—COMPOSITE ANOMALY CURVES FOR A PAIR OF POSITIVE LINE POLES FOR VARIOUS DEPTHS.

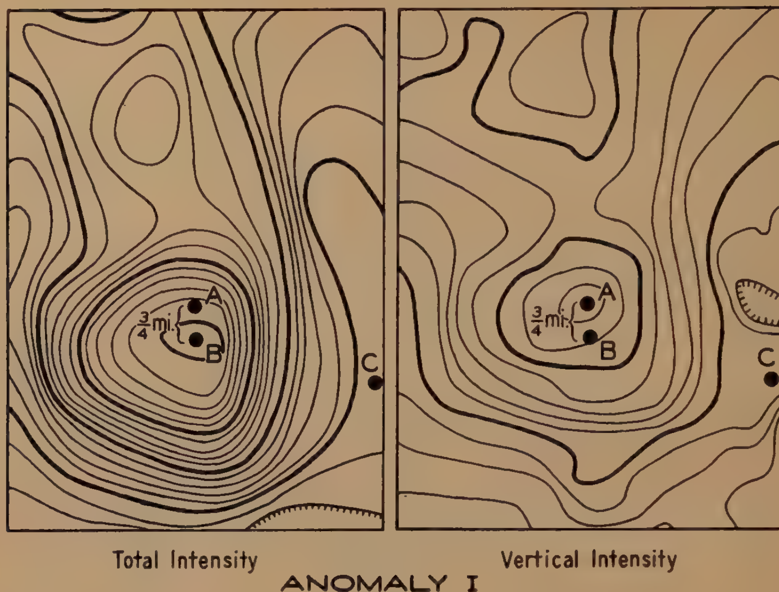

Fig. 7*a*

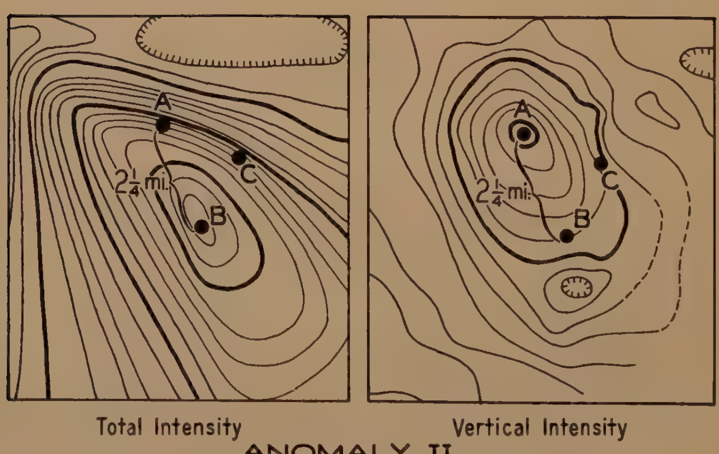

Fig. 7*b*

FIGS. 7*a* and 7*b*.—TOTAL AND VERTICAL INTENSITY ISOGAM MAPS FOR TWO OBSERVED ANOMALIES.

tion would be negative and the normal vertical field would be negative. This combination has not been studied sufficiently to warrant inclusion in the present paper (see Appendix, note 3) beyond the one curve included in Fig. 2*d*.

APPENDIX

1. A positive anomaly is one that corresponds to an attraction on a north-seeking pole. To be definite, we use the following convention:

A. A positive anomaly is due to a south-seeking pole.

B. The test pole is a north-seeking pole.

With this convention, the field of the earth induces a magnetic dipole in a magnetizable body, with a south-seeking pole near the upper south end and a north-seeking pole near the lower north end. For a negative anomaly the analysis becomes more complex. For the southern hemisphere, the analysis must also be revised.

2. Mathematically, the variables for the general case are:

A. The position of the test point (three variables).

B. The sign of the test pole (one variable).

C. The magnetic distribution of the disturbing body. (This involves a continuous range of four variables, one for the intensity and three for the point of the body.)

D. The normal field (three variables).

The number of variables can be reduced by idealization and arbitrary selections. Thus, the test pole may be taken as negative or north-seeking, eliminating the variable *B*. The body may be considered as consisting of a finite number of point or line poles. If there are n point poles, this involves $4n$ parameters, instead of a range of four variables. For n line poles, there are $5n$ parameters and $2n$ ranges of variables; the parameters may be taken as the three coordinates of some point on each line and the two directional parameters for the line, the ranges being for the magnetization and distance from a selected point on each line. If the disturbing body is taken as consisting of one horizontal line pole, lying perpendicular to the magnetic meridian and uniformly magnetized, the number of parameters is reduced to three for a fixed test point, one for the depth of the line, one for the distance of the test point magnetically north of the line pole and one for the strength of the pole. For a profile across the line pole, the horizontal distance becomes a variable. For definiteness, the analysis has been restricted to a positive anomaly in the north magnetic field of the earth, called "northern magnetic hemisphere" although its limits are not nearly so simple. In one figure (Fig. 2*d*) a curve is shown for a point at the magnetic equator and one for a southern magnetic hemisphere point, but these two curves are not considered in most of the discussion.

With these assumptions, the parameters are reduced to four;

A. The horizontal component H_0 of the normal undisturbed field, taken positive as it is always directed to the north.

B. The vertical component V_0 of the normal field, taken as positive when it is directed downward, as is assumed except for one curve in Fig. 2*d*.

C. The depth z of the body below the level of exploration. This may be the depth below the surface of the earth, as for surface measurements, or it may be the depth below the plane of flight, as for airplane measurements. It is taken as positive, although it might be negative in some applications, if the measurements were made below the level of the disturbing body.

D. The effective pole strength of the body, taken as positive by selection, although it may be negative for some actual bodies. For negative pole strengths,

the analysis becomes more involved although the calculations follow the same formulas.

There is one variable, x, the horizontal distance of the negative test point north of the vertical plane through the line pole. The sign of x is negative to the south of the pole and positive to the north. On the graphs, the south end of each profile is shown at the left.

3. The reader doubtless will think of many other combinations of parameters for which he would wish to have curves, or draw conclusions. The experience of the writer indicates that extension by analogy may lead to false conclusions because of the number of variables involved and because a polynomial often will change important characteristics with a change in the sign or numerical value of one or more of its coefficients. Limitations of time have precluded the preparation of an "album" of curves to cover all of even the major combinations. The purpose of the present paper has been to consider a few typical examples only and to indicate a few "pitfalls" in magnetic interpretations.

To facilitate computations for other choices of the parameters, Table 1 shows the vertical and the horizontal anomalies due to a horizontal, east-west line pole of strength $k = 1000$. In this table, depths are taken as $z = 1$ and $z = 2$, with values of x, the horizontal distance of the test point from the vertical plane of the pole, covering the range from 0 to 5 by steps of 0.1 and from 5 to 10 by steps of 0.5. Also, values of z are taken as 3, 4, and 5 with the values of x varying by steps of 0.5 from 0 to 5 and by steps of unity from 5 to 10.

In computation, the values of the anomaly components are adjusted to the desired strength and added algebraically to the assumed normal field components. These sums furnish the components of the total field. The "total intensity" is obtained, as usual, as the square root of the sum of the squares of the horizontal and vertical components. The "total anomaly" is the excess of the total intensity over the normal total intensity. The calculations are simple and direct but require considerable time and reliable curves call for judgment in the selection of points for calculation. The algebraic sign of V is positive above the pole, H is negative north of the pole and positive south of the pole.

As an example, suppose that

$$\begin{aligned} k &= 10{,}000 \\ H_0 &= 20{,}000 \\ V_0 &= 50{,}000 \\ z &= 2 \\ x &= 3.4 \end{aligned}$$

Then Table 1 gives:

$$H_a = \frac{10{,}000}{1000}(218.509) = 2185.09$$

and

$$V_a = \frac{10{,}000}{1000}(128.535) = 1285.35$$

$$\begin{aligned} \text{Hence: } H &= H_0 + H_a = 22185 \\ V &= V_0 + V_a = 51285 \\ T &= \sqrt{H^2 + V^2} = 55878 \\ T_0 &= \sqrt{H_0^2 + V_0^2} = 53852 \\ A &= T - T_o = 2026 \end{aligned}$$

If more accurate results are desired, the tables lead to:

$$\begin{aligned} H &= 22185.09 \qquad & V &= 51285.35 \\ T &= 55878.13 \qquad & T_0 &= 53851.65 \\ A &= 2026.48 \end{aligned}$$

As an example of multiple effects, suppose that there are two line poles; B_1 at depth 1 and distance 2 units south of the origin, with strength 15,000; B_2 at depth 2 and distance 3 units north of the origin, with strength 4000.

TABLE 1.—*Line Anomalies for Strength* $k = 1000$

$H_a = kx/(x^2 + z^2)$ $V_a = kz/(x^2 + z^2)$

$z = 1$			$z = 2$		
x	Horizontal	Vertical	x	Horizontal	Vertical
0.0	0	1000	0.0	0	500
0.1	99.010	990.099	0.1	24.938	498.753
0.2	192.308	961.538	0.2	49.505	495.050
0.3	275.229	917.431	0.3	73.350	488.998
0.4	344.828	862.069	0.4	96.154	480.769
0.5	400	800	0.5	117.647	470.588
0.6	441.176	735.294	0.6	137.615	458.716
0.7	469.799	671.141	0.7	155.902	445.434
0.8	487.805	609.756	0.8	172.414	431.034
0.9	497.238	552.486	0.9	187.110	415.800
1.0	500	500	1.0	200	400
1.1	497.738	452.489	1.1	211.132	383.877
1.2	491.803	409.836	1.2	220.588	367.647
1.3	483.271	371.747	1.3	228.471	351.494
1.4	472.973	337.838	1.4	234.899	335.570
1.5	461.538	307.692	1.5	240	320
1.6	449.438	280.899	1.6	243.902	304.878
1.7	437.018	257.069	1.7	246.734	290.276
1.8	424.528	235.849	1.8	248.619	276.243
1.9	412.148	216.920	1.9	249.671	262.812
2.0	400	200	2.0	250	250
2.1	388.170	184.843	2.1	249.703	237.812
2.2	376.712	171.233	2.2	248.869	226.244
2.3	365.660	158.983	2.3	247.578	215.285
2.4	355.030	147.929	2.4	245.902	204.918
2.5	344.828	137.931	2.5	243.902	195.122
2.6	335.052	128.866	2.6	241.636	185.874
2.7	325.694	120.627	2.7	239.150	177.148
2.8	316.742	113.122	2.8	236.486	168.919
2.9	308.183	106.270	2.9	233.683	161.160
3.0	300	100	3.0	230.769	153.846
3.1	292.177	94.251	3.1	227.774	146.951
3.2	284.698	88.968	3.2	224.719	140.449
3.3	277.544	84.104	3.3	221.625	134.318
3.4	270.701	79.618	3.4	218.509	128.535
3.5	264.151	75.472	3.5	215.385	123.077
3.6	257.880	71.633	3.6	212.264	117.925
3.7	251.872	68.074	3.7	209.158	113.058
3.8	246.114	64.767	3.8	206.074	108.460
3.9	240.592	61.690	3.9	203.019	104.112
4.0	235.294	58.824	4.0	200	100
4.1	230.208	56.148	4.1	197.021	96.108
4.2	225.322	53.648	4.2	194.085	92.421
4.3	220.626	51.308	4.3	191.196	88.928
4.4	216.110	49.116	4.4	188.356	85.616
4.5	211.765	47.059	4.5	185.567	82.474
4.6	207.581	45.126	4.6	182.830	79.491
4.7	203.551	43.309	4.7	180.146	76.658
4.8	199.667	41.597	4.8	177.515	73.965
4.9	195.922	39.984	4.9	174.938	71.403
5.0	192.308	38.462	5.0	172.414	68.966
5.5	176	32	5.5	160.584	58.394
6.0	162.162	27.027	6.0	150	50
6.5	150.289	23.121	6.5	140.541	43.243
7.0	140	20	7.0	132.075	37.736
7.5	131.004	17.467	7.5	124.481	33.195
8.0	123.077	15.385	8.0	117.647	29.412
8.5	116.041	13.652	8.5	111.475	26.230
9.0	109.756	12.195	9.0	105.882	23.529
9.5	104.110	10.959	9.5	100.796	21.220
10.0	99.010	9.901	10.0	96.154	19.231

$z = 3$		
x	Horizontal	Vertical
0.0	0	333.333
0.5	54.054	324.324
1.0	100	300
1.5	133.333	266.667
2.0	153.846	230.769
2.5	163.934	196.721
3.0	166.667	166.667
3.5	164.706	141.176
4.0	160	120
4.5	153.846	102.564
5.0	147.059	88.235
6.0	133.333	66.667
7.0	120.690	51.724
8.0	109.589	41.096
9.0	100	33.333
10.0	91.743	27.523

$z = 4$		
x	Horizontal	Vertical
0.0	0	250
0.5	30.769	246.154
1.0	58.824	235.294
1.5	82.192	219.178
2.0	100	200
2.5	112.360	179.775
3.0	120	160
3.5	123.894	141.593
4.0	125	125
4.5	124.138	110.345
5.0	121.951	97.561
6.0	115.385	76.923
7.0	107.692	61.538
8.0	100	50
9.0	92.784	41.237
10.0	86.207	34.483

$z = 5$		
x	Horizontal	Vertical
0.0	0	200
0.5	19.802	198.020
1.0	38.462	192.308
1.5	55.046	183.486
2.0	68.966	172.414
2.5	80	160
3.0	88.235	147.059
3.5	93.960	134.228
4.0	97.561	121.951
4.5	99.448	110.497
5.0	100	100
6.0	98.361	81.967
7.0	94.595	67.568
8.0	89.888	56.180
9.0	84.906	47.170
10.0	80	40

Let the test point be on the surface 0.3 units south of the origin. If the subscript 1 refers to B_1 and the subscript 2 refers to B_2, for the same normal field, as in the previous example:

$$x_1 = +1.7 \qquad x_2 = -3.3$$
$$z_1 = 1 \qquad z_2 = 2$$
$$k_1 = 15{,}000 \qquad k_2 = 4{,}000$$
$$H_0 = 20{,}000 \qquad V_0 = 50{,}000$$

Accordingly

$$H_1 = -15(437.018) = -6{,}555.270$$
$$V_1 = +15(257.069) = +3{,}856.035$$
$$H_2 = +\ 4(221.625) = +\ 886.500$$
$$V_2 = +\ 4(134.318) = +\ 537.272$$
$$H = H_0 + H_1 + H_2 = 14{,}332$$
$$V = V_0 + V_1 + V_2 = 54{,}393$$
$$T = 56{,}249$$
$$A = +2397$$

Magnetic Anomaly of Inclined Vein of Infinite Length

By Th. Koulomzine,* Member AIME and L. Massé†

(New York Meeting, March 1947)

Note on Haalck's Formula

Quantitative interpretation of magnetic anomalies is admittedly a difficult process. Few authors have attempted a general approach to this problem. A number of publications can be found widely scattered through scientific periodicals, which present particular solutions obtained by the introduction of simplifying assumptions, such as the reduction to an equivalent magnet or pole, or the hypothesis of a very thin vein. The practical applications of these solutions are necessarily very restricted though they have given satisfactory results in isolated cases.

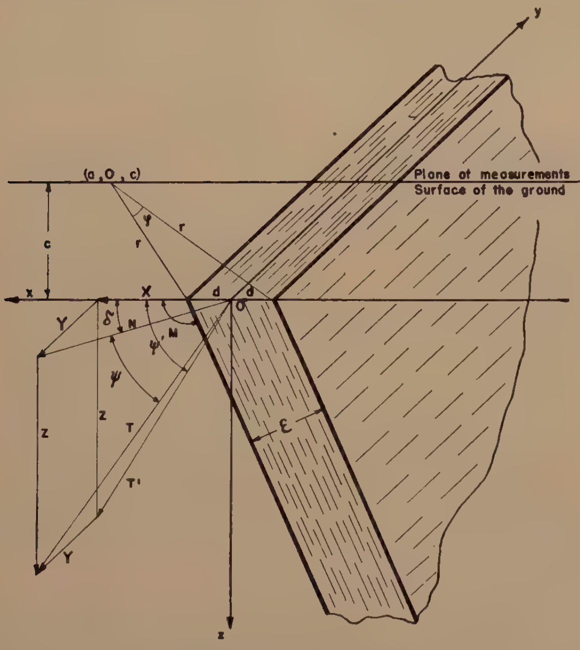

Fig 1—Volume element of coordinates introduced into magnetic field of total intensity with certain components

Probably the most widely known equation of this type was given by Haalck.[1] His classic work contains curves calculated for typical conditions, and has now become a standard reference.

However, a close scrutiny of Haalck's derivation reveals that, aside from the hypothesis of a thin vein, it rests on the

Manuscript received at the office of the Institute March 24, 1947. Issued as TP 2260 in Mining Technology, November 1947.

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[1] References are at the end of the paper.

separation theorem of multiple definite integrals which can not be used justifiably in this case.

To show this, we will reproduce briefly Haalck's reasoning using the notation that will be adopted throughout this paper.

Let the volume element dv of coordinates x, y, z be introduced into a magnetic field of total intensity T with components X, Y, Z (see Fig 1). It becomes magnetized by induction and its magnetic moment is

$$dM = kT\,dv, \qquad [1]$$

where k is the magnetic susceptibility appropriate to dv.

The Newtonian potential at a point of coordinates (a, b, c) is

$$W = \frac{1}{R}, \qquad [2]$$

where

$$R^2 = (x - a)^2 + (y - b)^2 + (z - c)^2 \qquad [3]$$

and, by Poisson's law, the magnetic potential is

$$dP = \frac{\partial W}{\partial s}\,dM; \qquad [4]$$

$$dP = \frac{k}{R^3}\,[X(x - a) + Y(y - b) + Z(z - c)]\,dv. \qquad [5]$$

The total potential at point (a,b,c) will be given by the triple integration of this result over the whole of the body creating the anomaly.

Consider an infinitely long vein striking in the y-direction (east-west), x being taken horizontally positive to the north and z vertically and positive downwards. We will take X to be the horizontal component of the magnetic field T in the plane perpendicular to the vein.

We must first carry out the integration in the y-direction,

$$P = k \int\!\!\int dx \cdot dz \int_{-\infty}^{\infty} \frac{X(x - a) + Y(y - b) + Z(z - c)}{R^3}\,dy. \qquad [6]$$

At this point Haalck assumed that, by symmetry, since the vein is infinitely long in the y-direction, the integration with respect to y can be replaced by a constant. This is mathematically equivalent to the application of the following theorem:

If a double integral (or generally, a multiple integral)

$$J = \int_{x_1}^{x_2}\int_{y_1}^{y_2} f(x,y)\,dx\,dy \qquad [7]$$

has constant limits of integration and an integrand in the form of the product of two functions, one of x alone, the other of y alone,

$$f(x,y) = g(x) \cdot h(y), \qquad [8]$$

then it can be written

$$J = \int_{x_1}^{x_2} g(x)\,dx \cdot \int_{y_1}^{y_2} h(y)\,dy.$$

This, clearly, is not the case of Eq 6 since the integrand is not separable as in Eq 8

The following change in variables

$$\tan\theta = \frac{y - b}{[(x - a)^2 + (z - c)^2]^{1/2}}$$

yields

$$P = 2k \int\!\!\int \frac{X(x - a) + Z(z - c)}{(x - a)^2 + (z - c)^2}\,dx\,dz. \qquad [9]$$

Duhoux[5] has carried out the somewhat tedious integration and has obtained for the vertical anomaly the following result

$$\Delta Z = \frac{2km}{1 + m^2}\left\{(mX - Z)\,\mathrm{Log}\sqrt{\frac{(a - d)^2 + c^2}{(a + d)^2 + c^2}} + (X + mZ)\left(\tan^{-1}\frac{-c}{a - d} - \tan^{-1}\frac{-c}{a + d}\right)\right\}, \qquad [10]$$

where $\qquad m = \tan M.$

For a thin vein, this reduces to

$$\Delta Z = -2k\epsilon T \frac{c \cos (M - \psi) + a \sin (M - \psi)}{R^2}, \qquad [11]$$

where ψ is the angle of dip of the magnetic field in the plane perpendicular to the vein and ϵ is the actual thickness of the vein. When the measuring instrument is above the top of the vein, as it usually is, c is negative.

For comparison, Haalck's equation is reproduced here. The notation has been changed to agree with the above.

$$\Delta Z = \frac{C \cdot Z}{R^3(a \sin M + c \cos M)^2} \Big\{ R^2 \cos M(a + R \cos M) + ac(a \sin M + c \cos M) - \frac{X}{Z}(a^3 \sin M + R^3 \sin M \cos M - c^3 \cos M) \Big\}. \qquad [12]$$

The original error was discovered by P. R. Geoffroy and the senior author some 17 years ago. A complete solution was worked out and a detailed study was made by the senior author of the singular points of some vertical anomaly curves. Unfortunately, the triple integration and the study of the critical points led to such complicated calculations that the work was never prepared for publication. Meanwhile, correct solutions have been published by Heiland,[2] Duhoux,[5] Evrard,[6] and others.[3,4]

New Derivation of Exact Formula

A more detailed mathematical study of the problem was undertaken by the junior author. This led to an entirely different approach that does away with the tedious triple integration and which can be applied to any two-dimensional problem. The new method of calculation has been suggested by the work of E. G. Kogbetliantz[7] on quantitative interpretation. The principles of the latter's method are reproduced here in an effort to show how easily and directly the desired equations are obtained. It will also become apparent that this method leads naturally to a simple way of computing vertical and horizontal components of the magnetic anomalies.

In 1824, Poisson established the law giving the magnetic potential of a body in a magnetic field, which we can write in vector form as follows:

$$P = \vec{I} \cdot \operatorname{grad} W \qquad [13]$$

where $\vec{I} = k\vec{T}$ is the intensity of magnetization in the magnetic field $\vec{T}$.

$$W = \iiint_V \frac{dv}{R} \qquad [14]$$

is the Newtonian potential.

Let x, y, z be the running coordinates and a, b, c the coordinates of the point where the potential is to be investigated.

We have, by definition

$$\operatorname{grad} W = \frac{\partial W}{\partial a}\vec{i} + \frac{\partial W}{\partial b}\vec{j} + \frac{\partial W}{\partial c}\vec{k} = A\vec{i} + B\vec{j} + C\vec{k}, \qquad [15]$$

where $\vec{i}, \vec{j}, \vec{k}$ are the usual triple orthogonal unit vectors.

Since the vein is assumed infinitely long in the y-direction, the magnetic field must be the same in every plane perpendicular to the y-axis. Eventually, it should prove to be a function of a and c only; that is, our problem is two-dimensional.

Assume the potential $W(a_0, b_0, c_0)$ at any fixed point, then

$$W(a,c) = W(a_0,c_0) - \iint_S dS \int_{-\infty}^{\infty} \left(\frac{1}{R_0} - \frac{1}{R}\right) dy, \qquad [16]$$

where

$$R^2 = (a - x)^2 + y^2 + (c - z)^2$$
$$R_0^2 = (a_0 - x)^2 + y^2 + (c_0 - z)^2.$$

If r and r_0 are the values of R and R_0 when $y = 0$, then

$$\int_{-\infty}^{\infty} \left(\frac{1}{R_0} - \frac{1}{R}\right) dy = 2\int_0^{\infty} \frac{dy}{\sqrt{y^2 + r_0^2}} - 2\int_0^{\infty} \frac{dy}{\sqrt{y^2 + r^2}}$$
$$= 2 \operatorname{Log}\left[\frac{y + \sqrt{y^2 + r_0^2}}{y + \sqrt{y^2 + r^2}}\right]_0^{\infty} = 2 \operatorname{Log} \frac{r}{r_0}$$

so that Eq 16 becomes

$$W = -2 \int\int_S \operatorname{Log} r \cdot dS + \text{constant}, \quad [17]$$

the integration to be extended to the cross-sectional area of the body causing the anomaly.

The magnetic anomaly will be given by

$$\Delta\vec{T} = \operatorname{grad} P = \operatorname{grad} (\vec{I} \cdot \operatorname{grad} W). \quad [18]$$

Using the well known equation

$$\operatorname{grad} (\vec{u} \cdot \vec{v}) = (\vec{v} \cdot \operatorname{grad})\vec{u} + (\vec{u} \cdot \operatorname{grad})\vec{v}$$
$$+ \vec{v} \wedge \operatorname{curl} \vec{u} + \vec{u} \wedge \operatorname{curl} \vec{v} \quad {}^*[19]$$

together with the identity

$$\operatorname{curl} \operatorname{grad} W \equiv 0$$

and the fact that $\vec{I}$ can be assumed constant, Eq 18 becomes

$$\Delta\vec{T} = (\vec{I} \cdot \operatorname{grad}) \operatorname{grad} W \quad [20]$$
$$= \left(I_a \frac{\partial}{\partial a} + I_c \frac{\partial}{\partial c}\right)(A\vec{i} + C\vec{k})$$
$$= (I_a W_{aa} + I_c W_{ac})\vec{i}$$
$$+ (I_a W_{ac} - I_c W_{aa})\vec{k}, \quad [21]$$

where

$$W_{aa} = \frac{\partial^2 W}{\partial a^2}, \quad W_{ac} = \frac{\partial^2 W}{\partial a\, \partial c},$$

and W is given by Eq 17.

In actual applications, it is preferable to write the field in complex notation considering

$$t = a + ic \text{ and } w = x + iz, \quad [22]$$

with $i^2 = -1$, as two complex variables.

With this change, Eq 21 goes over into

$$\Delta T = \Delta X + i\Delta Z = Ie^{-i\psi}(W_{aa} + iW_{ac}). \quad [23]$$

Taking the complex conjugate of both sides, we obtain

$$\Delta X - i\Delta Z = Ie^{i\psi}(W_{aa} - iW_{ac}). \quad [24]$$

From Eq 17 we get

$$W_a = -2 \int\int_S \frac{a - x}{r^2}\, dS,$$
$$W_c = -2 \int\int_S \frac{c - z}{r^2}\, dS,$$

and

$$W_a - iW_c = -2 \int\int_S \frac{(a - x) - i(c - z)}{r^2}\, dS$$
$$= -2 \int\int_S \frac{dS}{t - w}. \quad [25]$$

Changing the double integral to a contour integral by the Riemann-Green theorem, Eq 25 becomes

$$W_a - iW_c = 2 \int_\Gamma \operatorname{Log}(t - w)\, dz,$$

and finally

$$W_{aa} - iW_{ac} = 2 \int_\Gamma \frac{dz}{t - w},$$

so that

$$\Delta X - i\Delta Z = 2Ie^{i\psi} \int_\Gamma \frac{dz}{t - w}, \quad [26]$$

where Γ is the contour formed by the boundary of the area S.

This equation, given by Kogbetliantz,[7] can be used in all two-dimensional problems.

Consider the case of an inclined vein as shown in Fig 2.

The integration contour can be divided into three straight lines $\Gamma_1\Gamma_2\Gamma_3$. Along Γ_2

* $\wedge$ is Marco-Longo sign of a vector product.

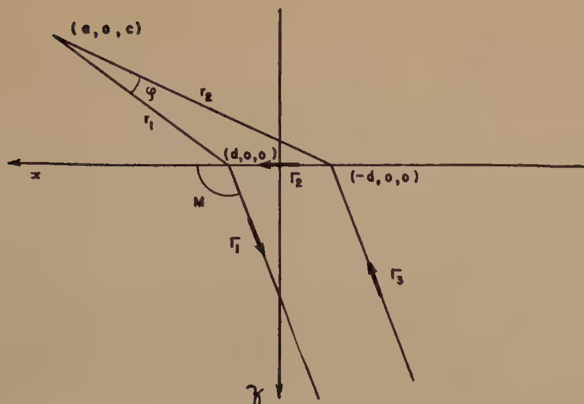

FIG 2—INCLINED VEIN.

$z = 0$ and $dz = 0$ so that the integral is zero. Thus, any horizontal discontinuity contributes nothing to the anomaly.

On Γ_1 we have $z = m(x - d)$, $w = d + \frac{1 + im}{m} z$, and the integral becomes

$$\int_0^\infty \frac{dz}{t - d - \frac{1+im}{m} z} = -\frac{m}{1 + im} \operatorname{Log}\left(t - d - \frac{1+im}{m} z\right)_0^\infty$$

$$= -\frac{m}{1+im} \operatorname{Log} \frac{t - d - \frac{1+im}{m} Z_0}{t - d}\Bigg]_{Z_0 \to \infty}. \qquad [27]$$

On Γ_3 we change d into $-d$ so that

$$\int_\infty^0 \frac{dz}{t + d - \frac{1+im}{m} z} = -\frac{m}{1 + im} \operatorname{Log}\left(t + d - \frac{1+im}{m} z\right)_\infty^0$$

$$= -\frac{m}{1+im} \operatorname{Log} \frac{t + d}{t + d - \frac{1+im}{m} Z_0}\Bigg]_{Z_0 \to \infty}. \qquad [28]$$

For the total integral, we have

$$-\frac{m}{1+im}\left\{\operatorname{Log}\frac{t+d}{t-d} + \operatorname{Log}\frac{t - d - \frac{1+im}{m} Z_0}{t + d - \frac{1+im}{m} Z_0}\right]_{Z_0 \to \infty} \qquad [29]$$

$$= \frac{m}{1+im} \operatorname{Log} \frac{t-d}{t+d}. \qquad [30]$$

Substituting this result into Eq 26 we get for the anomaly

$$\Delta X - i\Delta Z = 2Ie^{i\psi} \frac{m}{1+im} \operatorname{Log} \frac{t-d}{t+d}, \text{ or} \qquad [31]$$

$$\Delta X - i\Delta Z = \frac{2Im}{1+m^2} (\cos\psi + i \sin\psi)(1 - im) \operatorname{Log} \frac{t-d}{t+d}. \qquad [32]$$

But

$$\operatorname{Log}\frac{t-d}{t+d} = \operatorname{Log}\frac{(a-d)+ic}{(a+d)+ic}$$

$$= \operatorname{Log}\sqrt{\frac{(a-d)^2+c^2}{(a+d)^2+c^2}} + i\left[\tan^{-1}\frac{c}{a-d} - \tan^{-1}\frac{c}{a+d}\right]. \quad [33$$

To simplify the writing, let

$$\operatorname{Log}\sqrt{\frac{(a-d)^2+c^2}{(a+d)^2+c^2}} = \operatorname{Log}\frac{r_1}{r_2} = U$$

and

$$\tan^{-1}\frac{c}{a-d} - \tan^{-1}\frac{c}{a+d} = V.$$

Note that

$$\frac{m}{1+m^2} = \sin M \cos M.$$

Thus we have

$$\begin{aligned}\Delta X - i\Delta Z &= 2I \sin M \cos M \\ &\quad (\cos\psi + i\sin\psi)(1-im)(U+iV) \\ &= 2I\sin M\cos M\{[\cos\psi(U+mV) \\ &\quad -\sin\psi(V-mU)] + i[\sin\psi(U+mV) \\ &\quad +\cos\psi(V-mU)]\}. \quad [34]\end{aligned}$$

The vertical anomaly is given by the imaginary part of Eq 34.

$$\begin{aligned}\Delta Z &= -2I\sin M[\sin\psi\cos M(U+mV) \\ &\quad +\cos\psi\cos M(V-mU)] \\ &= 2I\sin M\left\{\operatorname{Log}\frac{r_1}{r_2}\sin(M-\psi)\right. \\ &\quad \left.+\varphi\cos(M-\psi)\right\}, \quad [35]\end{aligned}$$

where $\varphi = -V.$

Eq 35 agrees with Duhoux's[5] result given in Eq 10.*

In the same manner the horizontal anomaly is found from the real part of Eq 34 to be

$$\begin{aligned}\Delta X &= 2I\sin M\left\{\operatorname{Log}\frac{r_1}{r_2}\cos(M-\psi)\right. \\ &\quad \left.-\varphi\sin(M-\psi)\right\}. \quad [36]\end{aligned}$$

For a vertical vein, these equations become

$$\Delta Z = 2I\left(\operatorname{Log}\frac{r_1}{r_2}\cos\psi + \varphi\sin\psi\right) \quad [37]$$

$$= 2I\left(\operatorname{Log}\frac{r_1}{r_2}\sin\psi - \varphi\cos\psi\right). \quad [38]$$

* The equivalence of Eqs. 10 and 35 can be verified by substituting

$$\operatorname{Log}\frac{r_1}{r_2} = \operatorname{Log}\sqrt{\frac{(a-d)^2+c^2}{(a+d)^2+c^2}}$$

Also $kT = I$; $m\cos M = \sin M$; $X = T\cos\psi$; $Z = T\sin\psi$.

The preceding equations were obtained on the assumption that the strike of the vein was perpendicular to the magnetic meridian. It is easily generalized to the case where the normal plane to the vein makes an angle δ with the magnetic meridian.

In this case, it is necessary only to replace T by T', its projection on the normal plane, and ψ by ψ', the dip of the magnetic field as measured in the normal plane. These values are given by:

$$T' = T\cos\psi\sqrt{\cos^2\delta + \tan^2\psi}$$

$$\tan\psi' = \tan\psi\cdot\sec\delta$$

Construction of Circle Diagram

It is easy to appreciate, from Eq 35 and the following, the amount of calculations needed to draw a single anomaly profile. It must be remembered that all logarithms are in the natural system with base $e = 2.71828$ and the angles φ are expressed in radians. The process of interpretation would be greatly simplified if some rapid method of computation of the anomaly curves could be devised.

One such method is suggested by Eq 33 when written as follows

$$\operatorname{Log}\frac{t-d}{t+d} = U + iV \quad [39]$$

This constitutes a conformal mapping of the (U, V) plane on the t-plane that is used in electrostatics.

Writing

$$g = -2i\cot^{-1}\frac{it}{d},$$

so that

$$\cot\frac{ig}{2} = \frac{it}{d} = \frac{\cos\frac{ig}{2}}{\sin\frac{ig}{2}}$$

$$= i\,\frac{e^{-g/2}+e^{g/2}}{e^{-g/2}-e^{g/2}},$$

TABLE 1—*Data for Construction of Circles*

U or V	U Group Center	U Group Radius	V Group Center	V Group Radius
0.01	100.0033	99.9983	99.9967	100.0000
0.02	50.0067	49.9967	49.9933	50.0000
0.03	33.3433	33.3283	33.3233	33.3333
0.04	25.0133	24.9933	24.9867	25.0063
0.05	20.1667	19.9917	19.9833	20.0080
0.06	16.6867	16.6567	16.6467	16.6778
0.07	14.3090	14.2741	14.2624	14.2980
0.08	12.5267	12.4867	12.4733	12.5141
0.09	11.1411	11.0961	11.0811	11.1260
0.10	10.0333	9.9834	9.9666	10.0170
0.11	9.1276	9.0726	9.0542	9.1091
0.12	8.3733	8.3134	8.2933	8.3535
0.13	7.7356	7.6707	7.6489	7.7143
0.14	7.1895	7.1196	7.0961	7.1664
0.15	6.7166	6.6417	6.6166	6.6916
0.16	6.3032	6.2234	6.1966	6.2767
0.17	5.9389	5.8541	5.8256	5.9109
0.18	5.6154	5.5257	5.4954	5.5857
0.19	5.3263	5.2316	5.1997	5.2949
0.20	5.0665	4.9668	4.9331	5.0335
0.21	4.8317	4.7271	4.6917	4.7971
0.22	4.6186	4.5090	4.4719	4.5823
0.23	4.4242	4.3097	4.2709	4.3863
0.24	4.2464	4.1269	4.0864	4.2070
0.25	4.0830	3.9586	3.9163	4.0420
0.26	3.9324	3.8032	3.7591	3.8898
0.27	3.7933	3.6591	3.6133	3.7491
0.28	3.6643	3.5252	3.4776	3.6185
0.29	3.5444	3.4004	3.3511	3.4971
0.30	3.4327	3.2839	3.2327	3.3839
0.31	3.3285	3.1747	3.1218	3.2780
0.32	3.2310	3.0723	3.0176	3.1789
0.33	3.1395	2.9760	2.9195	3.0860
0.34	3.0537	2.8853	2.8270	2.9986
0.35	2.9729	2.7996	2.7395	2.9163
0.36	2.8968	2.7187	2.6567	2.8387
0.37	2.8249	2.6420	2.5782	2.7653
0.38	2.7570	2.5693	2.5037	2.6960
0.39	2.6928	2.5002	2.4328	2.6303
0.40	2.6319	2.4346	2.3652	2.5679
0.41	2.5742	2.3720	2.3008	2.5087
0.42	2.5193	2.3124	2.2393	2.4524
0.43	2.4672	2.2554	2.1805	2.3988
0.44	2.4175	2.2010	2.1241	2.3477
0.45	2.3702	2.1490	2.0702	2.2990
0.46	2.3251	2.0991	2.0184	2.2525
0.47	2.2821	2.0513	1.9686	2.2080
0.48	2.2409	2.0054	1.9208	2.1655
0.49	2.2016	1.9614	1.8748	2.1248
0.50	2.1640	1.9190	1.8305	2.0858
0.51	2.1279	1.8783	1.7878	2.0484
0.52	2.0934	1.8391	1.7465	2.0126
0.53	2.0602	1.8013	1.7067	1.9781
0.54	2.0284	1.7648	1.6683	1.9450
0.55	1.9979	1.7297	1.6310	1.9132
0.56	1.9686	1.6957	1.5950	1.8826
0.57	1.9404	1.6629	1.5601	1.8531
0.58	1.9133	1.6311	1.5263	1.8248
0.59	1.8872	1.6004	1.4935	1.7974
0.60	1.8620	1.5707	1.4617	1.7710
0.61	1.8378	1.5419	1.4308	1.7456
0.62	1.8145	1.5140	1.4007	1.7211
0.63	1.7920	1.4870	1.3715	1.6974
0.64	1.7702	1.4607	1.3431	1.6745
0.65	1.7493	1.4352	1.3154	1.6524
0.66	1.7290	1.4105	1.2885	1.6310
0.67	1.7095	1.3865	1.2622	1.6103
0.68	1.6906	1.3631	1.2366	1.5904
0.69	1.6723	1.3404	1.2116	1.5710
0.70	1.6546	1.3183	1.1872	1.5523

TABLE 1—(*Continued*)

U or V	U Group Center	U Group Radius	V Group Center	V Group Radius
0.72	1.6210	1.2758	1.1402	1.5166
0.74	1.5895	1.2355	1.0952	1.4830
0.76	1.5599	1.1972	1.0521	1.4515
0.78	1.5321	1.1607	1.0109	1.4219
0.80	1.5059	1.1260	0.9712	1.3940
0.82	1.4813	1.0929	0.9331	1.3677
0.84	1.4581	1.0612	0.8964	1.3429
0.86	1.4363	1.0309	0.8609	1.3195
0.88	1.4156	1.0014	0.8267	1.2975
0.90	1.3961	0.9742	0.7936	1.2766
0.92	1.3776	0.9475	0.7615	1.2569
0.94	1.3601	0.9219	0.7303	1.2383
0.96	1.3436	0.8973	0.7001	1.2207
0.98	1.3279	0.8737	0.6707	1.2041
1.00	1.3130	0.8509	0.6421	1.1884
1.1	1.2492	0.7487	0.5089	1.1221
1.2	1.1995	0.6625	0.3888	1.0729
1.3	1.1605	0.5888	0.2776	1.0378
1.4	1.1295	0.5251	0.1725	1.0148
1.5	1.1048	0.4696	0.0709	1.0025
1.6	1.0850	0.4210	−0.0292	1.0004
1.7	1.0691	0.3780	−0.1299	1.0084
1.8	1.0562	0.3399	−0.2333	1.0269
1.9	1.0458	0.3060	−0.3416	1.0567
2.0	1.0373	0.2757	−0.4577	1.0997
2.2	1.0249	0.2244	−0.7279	1.2369
2.4	1.0166	0.1829	−1.0917	1.4807
2.6	1.0111	0.1494	−1.6622	1.9397
2.8	1.0074	0.1221	−2.8127	2.9857
3.0	1.0050	0.0998	−7.0153	7.0829

we have

$$\frac{1+e^g}{1-e^g} = \frac{t}{d},$$

or

$$e^g = \frac{t-d}{t+d},$$

and

$$g = \operatorname{Log}\frac{t-d}{t+d}.$$

Eq 39 can now be written

$$\operatorname{Log}\frac{t-d}{t+d} = -2i\cot^{-1}\frac{it}{d} = U + iV.$$

From this we get

$$t = id\cot\frac{V-iU}{2}$$

$$= \frac{-d\sinh U + id\sin V}{\cosh U - \cos V}.$$

Since $t = a + ic$, we obtain

$$a = -\frac{d\sinh U}{\cosh U - \cos V}, \qquad [40]$$

$$c = \frac{d\sin V}{\cosh U - \cos V}. \qquad [41]$$

Eliminating first V and then U from Eqs 40 and 41, we arrive at the following equations, giving U and V as functions of a, c and d,

$$(a - d \coth U)^2 + c^2 = d^2 \operatorname{cosech}^2 U, \quad [42]$$

and

$$a^2 + (c - d \cot V)^2 = d^2 \operatorname{cosec}^2 V. \quad [43]$$

Eqs 42 and 43 show that the conformal mapping of the (U,V) plane on the (a,c) plane consists of a family of orthogonal circles.

Dividing both sides by d and writing

$$\alpha = \frac{a}{d} \text{ and } \gamma = \frac{c}{d},$$

the equations of the circles become

$$(\alpha - \coth U)^2 + \gamma^2 = \operatorname{cosech}^2 U \quad [44]$$

and $\alpha^2 + (\gamma - \cot V)^2 = \operatorname{cosec}^2 V. \quad [45]$

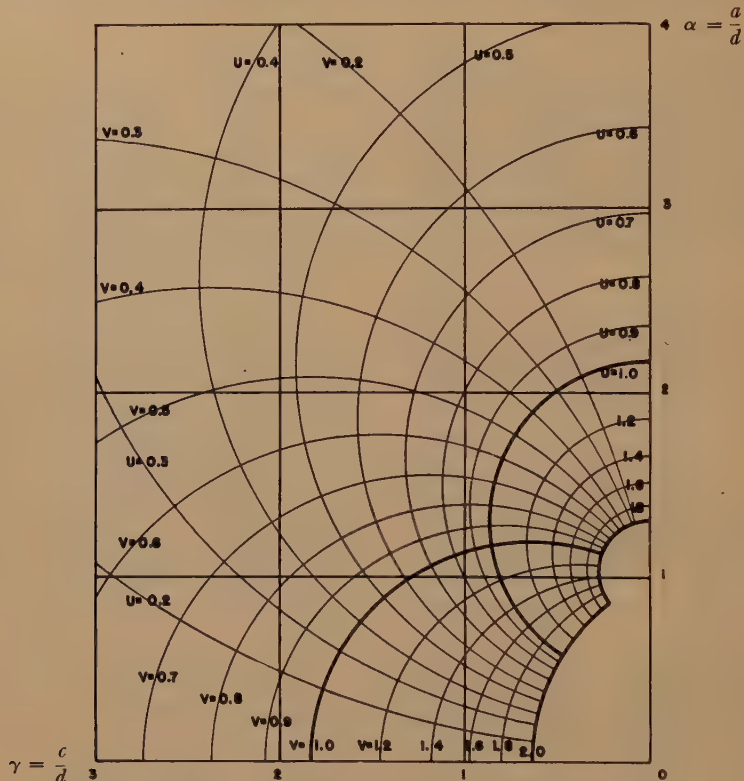

FIG 3—ORTHOGONAL CIRCLES FOR COMPUTATION OF ANOMALIES.

The family Eq 44 represents one circle for each value of U; its center is on the α-axis at a distance $\alpha = \coth U$ from the origin, and its radius is cosech U. Positive values of U correspond to negative values of α or a.

Similarly, one circle of Eq 45 corresponds to each value of V. Its center is on the γ-axis at a distance $\gamma = \cot V$ from the origin, and its radius is cosec V. V is positive for positive values of γ or c. All the circles of Eq 45 pass through the two points $\alpha = \pm 1$. On account of symmetry, it is only necessary to draw the circles for positive values of α and γ. A partial table of data necessary for the construction of the circles may be found in Table 1.

The circular nature of the curves makes the mapping very easy to construct, and

this is done once and for all (see Fig 3). For given values of a, c and d the corresponding values of U and V are immediately read off the graph and inserted into Eq 35.

As an application, consider the case of an E-W diabase dyke for which $k = 5.10^{-3}$ in a magnetic field $T = 6.10^4$.

Geometrical data:

depth	$c = -200$ ft
horizontal width	$2d = 200$ ft
dip of dyke	$M = 120°$ from north
dip of magnetic field	$\psi = 75°$.

Find the anomaly for $a = \pm 380$ ft.

Then, $\alpha = \pm 3.8$ and $\gamma = -2.0$.

From the graph, $U = \mp 0.458$ and $V = -0.321$.

Then for,

$a = -380$ ft (south)	$a = 380$ ft (north)
$0.71U = +0.325$	$0.71U = -0.325$
$-0.71V = +0.228$	$-0.71V = +0.228$
$+0.553$	-0.097

$2 \sin 120° = 1.73$, so that

$\Delta Z_s = +0.956kT$	$\Delta Z_N = -0.168kT$
$= 287\gamma$	$= -50\gamma$

Location of Maximum, Minimum and Inflection Points

For the purpose of determining the position of maximum, minimum and inflection points, it is convenient to rewrite Eq 35 as a function of α and γ instead of a, c, and d. We find

$$\Delta Z = 2I \sin M \left\{ \text{Log} \sqrt{\frac{(\alpha - 1)^2 + \gamma^2}{(\alpha + 1)^2 + \gamma^2}} \sin (M - \psi) + \cos (M - \psi) \left[\tan^{-1} \frac{-\gamma}{\alpha - 1} - \tan^{-1} \frac{-\gamma}{\alpha + 1} \right] \right\} \quad [46]$$

From this, the extreme values are located by

$$\frac{d(\Delta Z)}{d\alpha} = 0,$$

that is

$$\alpha^2 + 2\gamma\lambda\alpha - (1 + \gamma^2) = 0,$$

or

$$\alpha = -\gamma\lambda \pm \sqrt{1 + \gamma^2(1 + \lambda^2)}, \quad [47]$$

where

$$\lambda = \cot (M - \psi). \quad [48]$$

Thus the maximum occurs at

$$\begin{aligned} a_M = d\alpha_M &= d(-\gamma\lambda - \sqrt{1 + \gamma^2(1 + \lambda^2)}) \\ &= -c\lambda - \sqrt{d^2 + c^2(1 + \lambda^2)} \end{aligned} \quad [49]$$

and the minimum at

$$\begin{aligned} a_m = d\alpha_m &= d(-\gamma\lambda + \sqrt{1 + \gamma^2(1 + \lambda^2)}) \\ &= -c\lambda + \sqrt{d^2 + c^2(1 + \lambda^2)}. \end{aligned} \quad [50]$$

If $\lambda = \infty$ there is a maximum only at $a = 0$ and the curve is symmetrical with respect to this ordinate.

If $\lambda = 0$ the maximum and minimum are equidistant from the origin, the minimum being to the north.

Similarly, the inflection points are given by

$$\frac{d^2(\Delta Z)}{d\alpha^2} = 0,$$

or

$$\alpha^5 + 3\gamma\lambda\alpha^4 - 2(1 + \gamma^2)\alpha^3 + 2\gamma\lambda(\gamma^2 - 1)\alpha^2 + (1 - 2\gamma^2 - 3\gamma^4)\alpha - \gamma\lambda(1 + \gamma^2)^2 = 0. \quad [51]$$

This fifth degree equation cannot be solved in the general case. If numerical values for γ and λ are inserted, then particular solutions can be obtained numerically.

Some particular cases, however, are of interest.

The limiting case $c = 0$ yields

$$\alpha(\alpha^2 - 1)^2 = 0 \text{ except when } \lambda = \infty.$$

Thus we have a simple root at $\alpha = 0$ and two double roots at $\alpha = \pm 1$ except when $\lambda = \infty$ in which case these are simple roots.

When $\lambda = \infty$ (vein parallel to the magnetic field) Eq 51 reduces to

$$3\alpha^4 + 2(\gamma^2 - 1)\alpha^2 - (1 + \gamma^2)^2 = 0 \quad [52]$$

and there are only two inflection points symmetrical with respect to the center of the vein.

If $\lambda = 0$ (vein perpendicular to the magnetic field), Eq 51 becomes

$$\alpha^5 - 2(1 + \gamma^2)\alpha^3 + (1 - 2\gamma^2 - 3\gamma^4)\alpha = 0. \quad [53]$$

There is always an inflection point above the center of the vein. The others are given by

$$\alpha^2 = (1 + \gamma^2) \pm \sqrt{(1 + \gamma^2)^2 - (1 + \gamma^2)(1 - 3\gamma^2)}. \quad [54]$$

They are symmetrical with respect to the center of the vein.

It is apparent that if $1 - 3\gamma^2 > 0$ there are five inflection points in the anomaly curve, while if $1 - 3\gamma^2 < 0$ there are only three. The limiting value of γ is obtained from $1 - 3\gamma^2 = 0$

$$\gamma = \frac{\sqrt{3}}{3} = 0.57735.$$

Thus $$\frac{2d}{c} = \frac{2}{\gamma} = 2\sqrt{3} = 3.4641. \quad [55]$$

It will be shown later that the condition in Eq 55 holds for all finite values of λ.

Thus, if the vein is not parallel to the magnetic field and if its horizontal width is more than $2\sqrt{3}$ times its depth, there are five inflection points, otherwise there are not more than two or three.

Thin-vein Case, Review of Interpretation Methods

The case of the thin vein has been completely solved by Duhoux,[5] Evrard[6] and de Magnée and Raynaud.[8] Only the main results will be reproduced here for the sake of completeness.

Where the thickness of the vein ϵ is small compared with the depth of the top of the vein, the functions U and V can be expanded in powers of $1/R$ and only the first terms retained.

$$U = -\frac{2ad}{R^2} = -\frac{a\epsilon}{R^2 \sin M} \text{ and}$$

$$V = \frac{2cd}{R^2} = \frac{c\epsilon}{R^2 \sin M}.$$

These values substituted into Eq 35 yield Eq 11

$$\Delta Z = -2k\epsilon T \frac{c \cos (M - \psi) + a \sin (M - \psi)}{R^2}, \quad [11]$$

where $$R^2 = a^2 + c^2.$$

The positions of the maximum and minimum points are obtained from Eq 47 multiplying through by d and retaining the highest powers in c. Then

$$a^2 + 2c\lambda a - c^2 = 0,$$

or

$$a = -c(\lambda \pm \sqrt{1 + \lambda^2}),$$

so that the maximum is located at

$$a_M = -c\lambda - c\sqrt{1 + \lambda^2} = c \tan \frac{M - \psi}{2} \quad [56]$$

and the minimum at

$$a_m = -c\lambda + c\sqrt{1 + \lambda^2} = -c \cot \frac{M - \psi}{2} \quad [57]$$

The midpoint of this distance $(a_M a_m)$ is located at

$$a_0 = -c\lambda \quad [58]$$

Eq 11 shows that this point corresponds to zero anomaly. *This is a characteristic property of thin veins.*

The interpretation in the case of a thin vein can be carried out as follows:

A straight line DD' is drawn to represent the ground level (see Fig 4). Two points are marked on this line, corresponding to

a_m and a_M the minimum and maximum as determined from the field curve and, with the distance $a_m a_M$ as diameter, a circle is drawn with point a_0 as center. The center of the top of the vein should be somewhere on this circle.

On the anomaly curve itself, the points of maximum and minimum values are joined by a straight line cutting the anomaly curve at a point Q. It can be shown that a perpendicular from point Q to line DD', when extended will cut the circle at the center of the vein (point O). Eq 58 shows that the angle Oa_0D' is equal to $(M - \psi)$ which defines the dip of the vein.

The values of depth and dip thus determined can be used to compute the anomaly at any point. When compared with the actual value obtained in the field, this yields the product $k\epsilon$ of the susceptibility by the thickness of the vein. It is impossible to separate these two factors. If, however, from geological considerations, one of them can be assumed, the other, of course, becomes known.

There is one case where Duhoux's method fails: that is the symmetrical or nearly symmetrical case. Since no minimum appears on the curve or is so far removed as to be useless, it is obvious that the circle mentioned above cannot be drawn. This situation arises when the plane of the vein contains the magnetic-field vector, that is when $M = \psi$.

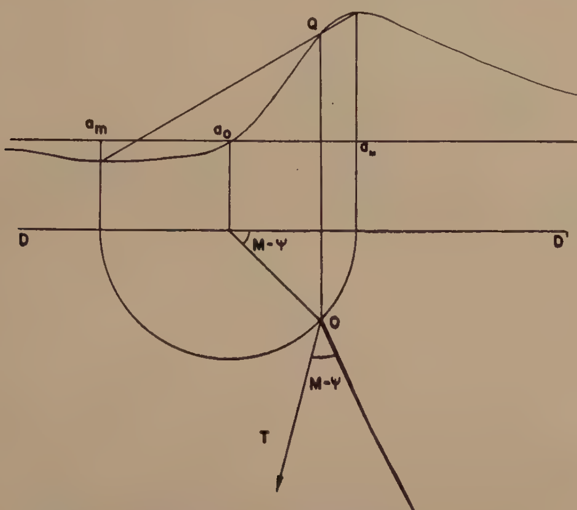

FIG 4—CHARACTERISTIC PROPERTY OF THIN VEINS.

The depth to the top of the vein can then be found by the rule first given by Rössiger and Puzicha:[10] the depth is equal to ½ the distance separating the points where the anomaly has fallen to ½ its maximum value. If we call $2l$ this distance, we have

$$c = l \cos (M - \psi) \qquad [59]$$

or very nearly $c = l$ when M is nearly equal to ψ. When $-20° < M - \psi < 20°$ the error introduced in the determination of c is less than 6 pct in excess of the actual value. This rule does not depend on the position of the minimum.

The thin-vein problem is thus completely solved except for the resolution of the product $k\epsilon$.

It is to be noted that the nature of this solution is fundamentally different from the one given by Lee.[9] As a general

rule, no equivalent magnet can be found for the thin vein.

Thick-vein Case, Determination of its Geometrical and Magnetic Parameters

The case of the thick vein is much more difficult and does not appear to have been solved until now. Quite a number of rules have been devised which all break down at one point or another. They may be used however, to give rapidly the upper bounds of certain parameters. The superposition method gives approximate information on all geometrical factors but necessitates a large number of graphs.

We have developed a method which, we feel confident, will solve the problem of interpretation in the case of every anomaly produced by an infinitely long dyke of infinite vertical extent. If the ratio of the width to the depth of the dyke is less than one, Duhoux's method will give a satisfactory approximation to the values of depth and dip.

The whole interpretation process is based on the location of the maximum, minimum and inflection points as given by Eqs 47 and 51. These equations are solved for different values of γ and λ; we obtain the maximum α_M, the minimum α_m and the inflection points f_1 f_2 f_3 f_4 f_5 numbered from the north. In the case $\gamma > 0.577$, f_2 and f_4 are complex conjugate roots; there are then only three inflection points. In the symmetrical case there are only two.

The fifth degree equation can be solved

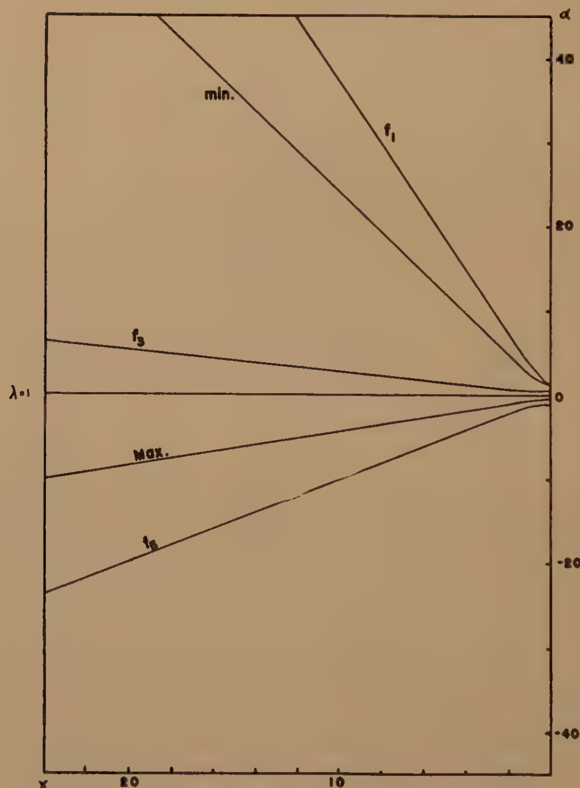

Fig 5—Property of thick veins.

by any of the standard methods of Wronski, Newton-Raphson, Bernouilli, Graeffe or Sturm.

For each of 18 values of the dip M (every 10°), a graph is constructed showing the values of α_M, α_m, f_1, f_3, f_5 for different values of γ, say from 0 to -20. Fig 5 shows such a graph for $\lambda = 1$ or $M - \psi = 45°$.

We can however, reduce the number of graphs needed to ten because of symmetry. It is obvious that if we multiply the successive terms of a polynomial by 1, K, K^2, K^3 and so on, beginning with the constant term, we obtain a new polynomial whose roots are K times those of the original one. Changing λ into $-\lambda$ in Eqs 47 and 51 is equivalent to taking $K = -1$. Thus the sign of all the roots is changed. It is then necessary to construct graphs for positive values of λ. It will be found that the curves tend very rapidly toward a straight line corresponding to the case of a thin vein.

We will first consider the general case where $\lambda \neq 0$, $\lambda \neq \infty$ (the vein is neither parallel nor perpendicular to the magnetic field).

The first step in the interpretation is to determine the position of the center of the top of the vein (that is, point O). This can be done accurately with the theory of moments. Since this method turns out to be somewhat involved, a graphical method can be substituted with satisfactory results. The process is as follows: On a sheet of transparent paper, a straight line is drawn on which the actual points a_M a_m $F_1 = df_1$ $F_3 = df_3$ $F_5 = df_5$ are plotted to any convenient scale. This sheet is then moved up and down on one of the graphs just described, keeping the line parallel to the axis of α until near coincidence is obtained. On the transparent sheet, the position $\alpha = 0$ is marked. This gives the center of the vein. The approximate value of λ can be read from the graph used to obtain the best coincidence though it will not be needed at this point but only to provide a check later. By careful drawing of tangents or by second difference tables the inflection points can be obtained with sufficient accuracy to locate point O to within a few feet.

Using this point O as new origin, all positions are recalculated. These new values will be understood in the theory that follows.

Consider Eq 51 as a fifth degree equation in x. Since there are three real roots in the general case, it can be written

$$(x + x_1)(x + x_2)(x + x_3)(x^2 + px + q) = 0.$$

Writing
$$S = x_1 + x_2 + x_3,$$
$$\Sigma = x_1x_2 + x_1x_3 + x_2x_3,$$
$$\Pi = x_1x_2x_3,$$

multiplying and identifying each term with Eq 51 after multiplication by 1, d, d^2, and so forth, and noting that

$$x_1 = -F_1, \quad x_2 = -F_3, \quad x_3 = -F_5,$$

we find

$$S + p = 3c\lambda, \tag{60}$$
$$\Sigma + pS + q = -2(c^2 + d^2), \tag{61}$$
$$\Pi + p\Sigma + qS = 2c\lambda(c^2 - d^2), \tag{62}$$
$$p\Pi + q\Sigma = (d^2 + c^2)(d^2 - 3c^2), \tag{63}$$
$$q\Pi = -c\lambda(c^2 + d^2)^2, \tag{64}$$

with
$$S = -(F_1 + F_3 + F_5),$$
$$\Sigma = F_1F_3 + F_1F_5 + F_3F_5,$$
$$\Pi = -F_1F_3F_5.$$

Eq 47 gives

$$a_M + a_m = -2c\lambda, \tag{65}$$
$$a_Ma_m = -(d^2 + c^2). \tag{66}$$

The five quantities a_M a_m F_1 F_3 F_5 are read directly off the anomaly curve and S Σ and Π computed.

The product $c\lambda$ determined from Eq 65 is substituted in Eq 60 to give p. This is used with Eqs 61 and 66 to give q. Finally, Eq 62 gives the difference of the squares $(c^2 - d^2)$ while the sum is known

from Eq 66. Thus both c and d are determined. Eq 65 then yields λ. Since ψ is known, the dip M can be calculated.

It can be shown that Eq 55 holds rigorously only when $p = 0$, but since p is always very small, the error in Eq 55 is negligible.

To determine the susceptibility, note that Eq 35 can be written, for any point a,

$$\Delta Z(a) = kAU(a) + kBV(a), \qquad [67]$$

where

$$A = 2T \sin M \sin (M - \psi), \qquad [68]$$
$$B = 2T \sin M \cos (M - \psi) \qquad [69]$$

are constants known from previous operations.

But $U(a)$ is an odd function and $V(a)$ an even function. We have then

$$\Delta Z(-a) = -kAU(a) + kBV(a), \qquad [70]$$

and

$$\Delta Z(a) - \Delta Z(-a) = 2kAU(a). \qquad [71]$$

$U(a)$ can easily be evaluated for any value of a and the computed values of c and d from the circle diagram. Thus the susceptibility is known.

The interpretation of the general case is then completed. We must now consider the two particular cases.

First, consider $\lambda = 0$. In this case we have

$$a_M = -a_m, \quad F_1 = -F_5, \quad F_3 = 0,$$
$$S = 0, \quad \Pi = 0, \quad \Sigma = -F_1^2.$$

Eq 66 becomes

$$a_M^2 = d^2 + c^2, \qquad [72]$$

while Eqs 60 and 61 give

$$p = 0, \quad q = F_1^2 - 2a_M^2, \qquad [73]$$

and Eq 63 is

$$q\Sigma = (d^2 - 3c^2)a_M^2. \qquad [74]$$

With Eqs 72, 73 and 74 the problem is completely solved. The center of the vein is located under the point of zero anomaly.

The case $\lambda = \infty$ (symmetrical case) does not seem to lend itself to complete solution by analytical means. However, the center of the vein is obviously under the maximum value. The dip is also known: the vein is parallel to the magnetic field. In this case the superposition method will give satisfactory results since the symmetrical curves vary rapidly with changing ratios γ.

The upper bound of the depth of the dyke can be obtained by the Rössiger and Puzicha rule. This upper bound is the exact depth of the thin vein case, the error increasing with the width. When γ is small, the sides of the vein are nearly at the points of inflection, since, when $\gamma = 0$, Eq 52 reduces to

$$3\alpha^4 - 2\alpha^2 - 1 = (3\alpha^2 + 1)(\alpha^2 - 1) = 0;$$
$$\alpha = \pm 1 \text{ or } a = \pm d.$$

If we assume the depth computed from Eq 59, the width of the dyke can be evaluated from Eq 52.

TABLE 2—*Vertical Anomaly over Thick Vein*

a	North	South
0	340.7	
10	309.2	367.6
20	274.1	386.7
30	236.8	404.1
40	198.7	413.1
50	161.2	415.9 max.
60	125.6	413.3
70	92.7	406.1
80	63.2	395.1
90	37.3	381.6
100	15.2	366.3
150	−50.2	286.6
200	−71.2	223.7
250	−75.2 min.	179.5
300	−73.2	148.3
350	−69.3	125.7
400	−64.9	108.7
450	−60.6	95.5
500	−56.6	85.1
600	−49.7	69.7
700	−44.1	58.9
800	−39.6	50.9
900	−35.9	44.9
1000	−32.8	40.0

$c = -100$ ft, $d = 50$ ft, $M = 120°$, $\psi = 75°$, $\delta = \infty$, $k = 5.10^{-4}$, $T = 6.10^4$

To illustrate the process and check the validity of the theory, a complete

example will be worked out. For this purpose, the anomaly curve corresponding to $\lambda = 1$, $c = -100$ ft, $d = 50$ ft has been computed for diabase $k = 5.10^{-3}$ and $T = 6.10^4$. If $\psi = 75°$ then the dip $M = 120°$ from the north. The results are given in Table 2 and plotted in Fig 6.

The exact positions of the critical points are as follows:

$$a_M = -50.0,\ a_m = 250.0,$$
$$F_1 = 381.3,\ F_3 = 35.5,\ F_5 = -113.4.$$

The interpretation has been carried out for seven cases for comparison. The first two used the exact and the rounded values of distances. The next four show the effect of rather large shifts in the position of the point O. The last method was Duhoux's approximation of a thin vein. The results will be found in Table 3.

With the exact values given above, we find

$$S = -301.4,\ \Sigma = -34{,}558,\ \Pi = 1{,}562{,}164.$$

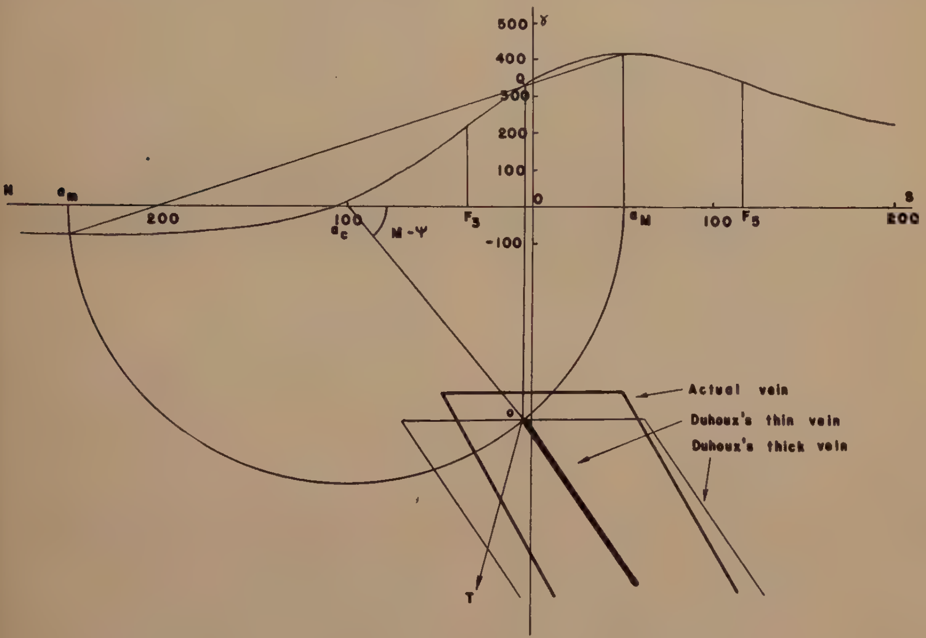

Fig 6—Application of Duhoux's method on low values.

Table 3—*Values Obtained in Interpretation of Seven Cases*

Point O	Correct	Correct	5 Ft to N	5 Ft to S	10 Ft to N	10 Ft to S	Thin Vein
a_M	−50.0	−50	−55	−45	−60	−40	
a_m	250.0	250	245	255	240	260	
F_1	381.3	380	375	385	370	390	
F_3	35.5	35	30	40	25	45	
F_5	−115.4	−115	−120	−110	−125	−105	
S	−301.4	−300	−285	−315	−270	−330	
Σ	−34,588	−34,425	−37,350	−31,350	−40,125	−28,125	
Π	1,562,164	1,529,500	1,350,000	1,694,000	1,156,250	1,842,750	
p	1.4	0	0	0	0	0	
q	10,003	9,425	10,400	8,400	11,325	7,325	
c	−100	−97	−105	−90	−112	−81	−116
d	50	54	50	59	44	62	66
λ	1.00	1.026	0.905	1.17	0.803	1.35	
$M - \psi$	45°	44°	48°	41°	51°	37°	51°

Then

$$2c\lambda = -(a_M + a_m) = -200,$$
$$c^2 + d^2 = -a_M a_m = 12{,}500,$$
$$p = 3c\lambda - S = -300 + 301.4 = 1.4,$$
$$pS = -414.4, \quad p\Sigma = -47{,}559,$$
$$q = -2(c^2 + d^2) - pS - \Sigma,$$
$$= -25{,}000 + 34{,}558 + 414 = 10{,}002,$$
$$qS = -3{,}014{,}505,$$
$$2c\lambda(c^2 - d^2) = \Pi + p\Sigma + qS,$$
$$= -1{,}499{,}900.$$

Hence

$$c^2 - d^2 = 7500, \text{ and } c^2 + d^2 = 12{,}500,$$

so that $c = -100$ ft, and $d = 50$ ft

$$\lambda = -\frac{(a_M + a_m)}{2c} = \frac{-200}{-200} = 1.00.$$

Then

$$\Delta Z(a) - \Delta Z(-a) = 2kU(a),$$

for $a = 100$ ft and the values just computed we find $U(100) = -0.4778$. And from Table 2:

$$\Delta Z(100) = 15.2, \quad \Delta Z(-100) = 366.3,$$

so that

$$2AU(100) = -70{,}092,$$

and

$$k = \frac{-351.1}{-70{,}092} = 5.01 \times 10^{-3}.$$

It will be appreciated that the method gives accurate results with relatively few computations when accurate data are provided. We are fully aware that field data are nowhere near this accuracy as a general rule, but we feel that, when the geological problem to be solved warrants it, repeated readings at closely spaced stations carefully corrected for disturbances will provide data with the degree of accuracy needed. If the resulting curve is smoothed out by known methods and the inflection points located by second difference tables, the error in the computed parameters can be kept very low.

The position of the center of the vein (although known in this case) was determined by the method described above. The transparent graph was prepared using three different scales. The errors were all less than 2 ft. In no instance should the error in the position of the point O exceed 5 ft either way; the accuracy should then be better than that shown in Table 3.

At the center of the distance $a_M a_m$, that is for $a = 100$ ft the anomaly is 15γ. Although we are not dealing with a thin vein, this value is low enough to suggest the application of Duhoux's method. This has been done in Fig 6, with the results entered in Table 3. The errors in this case are much higher than those found in our method.

The rule of Rössiger and Puzicha[10] in this case gives a depth of 125 ft. This error of 25 pct is larger than any tabulated.

Conclusions

This paper describes an easy method of computing anomalies caused by infinitely long dykes. It applies equally well to wide dykes and to thin veins. It is more accurate and faster than the graphical methods suggested by Duhoux and Lee. As such, it should appeal to the geophysicists who are engaged in solving geological and mining problems.

A second contribution of this paper concerns the interpretation of the thick dyke anomaly. The original assumption of infinite length and depth are, of course, never fulfilled in actual field practice. It cannot be deduced from this, however, that the accurate interpretation is impossible. In fact, every elementary volume will affect the instrument according to the inverse cube law, so that, in a relatively short distance, magnetic effects will become negligible. It is sufficient, for satisfactory results, that the vein-like structure under investigation be plane, of uniform section and homogeneous near the plane of the profile. Folding and faulting may occur without effect on the

readings at a distance ten times the depth of overburden.

The proposed method yields accurate results with a minimum of actual computation though it requires some care in field measurements. Where the geological problem to be solved is important, additional precautions are surely warranted.

The methods that have been proposed up to now give only approximate values for some of the parameters involved; the errors are considerable in some cases. Furthermore, none solve the problem completely.

We hope to publish, in the near future, a complete set of master curves to assist in the interpretation of vein-like structures.

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Geophysical Investigations for Selection of Site for Ramapadasagar Dam across the Godavari River in Madras, South India

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(New York Meeting, March 1947)

Abstract

This paper records the results of the earth resistivity surveys made in the Godavari River in connection with the Ramapadasagar project. After describing the topographical and geological features of the area, the results of the investigation at the several sites have been mentioned. Further, a detailed account has been given of the resistivity methods adopted, the apparatus used and the interpretation of the curves. The resistivity measurements carried out over the water course have been specially outlined, and a review of the verification of the electrical indications by drilling in a number of cases, has also been presented.

Introduction

The Government of Madras has under consideration the Ramapadasagar project which is estimated to cost some Rs.660,000,000 (about two hundred million dollars). The most essential feature of this project is the construction of a high dam across the Godavari near Polavaram for impounding the waters of this great river of South India, whose maximum discharge is 2 million cu secs in flood time. It is estimated that when the proposed reservoir is constructed, about one million tons of rice could be produced annually under irrigation. To those who have had any acquaintance at all with the recent distressing famine conditions which India has passed through and is still undergoing, the bounteous effects which this project will bring about to the country in the future, are obvious.

In the region of the proposed reservoir, the river flows for several miles through a hilly tract, hemmed in between flanks of hard, solid rock; but the river bed is mostly sandy. A few bore holes put down in this sandy bed touched rock at a depth of 180 to 200 ft. The river bed is about 50 ft above the mean sea level (m.s.l.) and the bedrock was found at −125 to −160 ft m.s.l. The crest of the dam is proposed at +162 ft m.s.l. The proposed dam would therefore be about 120 to 130 ft above the river bed, and the foundations would have to reach down nearly 200 ft below the bed at the deepest part. A number of alternative sites were proposed for investigation, and it became necessary to quickly ascertain the bedrock topography at these sites. Dr. M. S. Krishnan,* who was consulted in the matter, furnished a detailed account of the geological features of the region and suggested a geophysical investigation which the writer carried out from February to April 1944, employing solely the electrical resistivity methods.

The total length of the survey lines for the electrical measurements amounted to 62,600 ft and the number of resistivity determinations was about 2500. Altogether nine alignments for the dam were inten-

Published with the permission of the Government of Madras. Manuscript received at the office of the Institute Jan. 17, 1947. Issued as TP 2287 in Mining Technology, November 1947.

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sively examined at five different sites. The engineers naturally wanted us to find out, if possible, a site wherein the dam alignment could be so taken as to secure hard rock throughout, at or above −60 ft m.s.l., or, within 100 ft below the ground surface. They considered that at such a depth they could easily find the masonry without having recourse to the expensive caissons or coffer dams. The geophysical investigation, however, soon indicated that the erosion of the bedrock has been so profound in the area that a deep section with rock at a depth of about 200 ft in the river bed was unavoidable in all the alignments. However, below the sands near Ramayyapeta a favorable attitude of bedrock, which involved a considerably narrowed deep section, was discovered by the resistivity surveys and a new alignment was selected taking advantage of this feature. Extensive drilling was proposed on this site and the results obtained have amply confirmed the geophysical indications.

A brief account of the earth-resistivity surveys carried out in the area, the apparatus employed, the methods of interpretation adopted and a review of the verifications by the borings will be presented.

Topography

The Godavari River, after flowing through a winding gorge across the range of rugged hills known as the Eastern Ghats, between lat. 17° 15′ to 30′ N. and long. 81° 15′ to 27′ E., emerges into the plains near Polavaram. Soon after leaving Polavaram, the river widens out enormously to two to three miles or more across its banks, branching off into smaller streams separated by sandy islands, and gradually merging into the deltaic region. In the hilly tract above Polavaram, for about 15 miles, the river has an average width of about one mile with several constrictions and expansions, until higher up in the jungle country where it flows in a very deep, narrow, magnificent gorge, just about a furlong wide between its precipitous rocky cliffs. This gorge is highly renowned for its wild, picturesque scenery.

The narrow sections higher up in the interior of the hilly tract are not suitable for the construction of the dam since the off-take of water from those portions would be exceedingly costly, involving numerous tunnels and deep cuttings along miles of country. Consequently, for the purpose of the project, only the small region a few miles above Polavaram is regarded as suitable. The river section in this short stretch of suitable country consists of the following portions:

1. The permanent stream (or the so-called summer water course) is 50 to 60-ft deep in many places and occasionally more than 100 ft deep. The top level of this water course is usually about +40 ft m.s.l. and a considerable part of the bottom is lower than sea level.

2. The sandy bed is a conspicuous feature forming an immense spread either to the left or to the right side of the stream, depending upon the convexity of the meanders. The average level of this sandy bed is at +50 to +60 ft m.s.l.

3. The high level terrace (or the flood plain) is always about 20 to 40 ft above the bed of the river, rising in a cliff bank with ramps near the villages for approaching the water course. This alluvial terrace may be present either on one or both the banks; it may border the water course directly, or a sandy bed may intervene.

4. The hills flanking the river bed attain elevations up to 1200 ft. Occasionally, a small isolated mound rising to about a hundred feet above the sands and forming a rocky island is found in the river bed.

Geology

The geological formations of the area consist of: (1) the Archaean gneisses which constitute the hills and also form the bedrock; and (2) the Recent alluvial deposits filled in by the river.

The Archaean gneisses comprise garnetiferous gneisses, sillimanite gneisses, leptynites and charnockites. These types form a complex of highly compact, banded and sometimes coarsely granitic rocks, and have been regarded as highly metamorphosed and granitized ancient sediments.

Structurally, the rocks are all quite compact and hard, although banded and foliated in places. The strike of the banding and foliation is usually east-west, swerving locally east-northeast to northeast. The dip throughout is southerly, at high angles, usually 60° to 70°. The rocks as a whole constitute a massive crystalline formation, and as such they are strong and sound enough to bear the weight of the dam. The main consideration in choosing a site is as Dr. Krishnan put it, "the depth to bedrock, the length of dam involved by the alignment, and the cost of construction of channels from the reservoir to the plains below." To appreciate the factors governing the depth to bedrock in the area, we may turn our attention to the Recent alluvial deposits.

These river deposits consist mostly of sands, clays and pebbles. The sands are usually very coarse and highly garnetiferous. Their total thickness is appreciable, being more than 200 ft in many places, as revealed by the geophysical indications and the bore holes put down in the bed of the river for the investigations of the project. Intercalated with the sands are clays and also pebble beds which are very irregular, and comparatively speaking, quite thin. The clays attain a thickness of about 100 ft or so only in the high level terraces or the flood plains, and in such cases, they usually are underlaid by a lower horizon of sands 100 to 120-ft thick, above the bedrock. The river apparently has eroded the rocks quite deeply and has filled up the depression made by the alluvial materials borne by it in its later stages.

As stated before, the rocky bed of this river lies generally about 125 to 150 ft below sea level and in a narrow section probably even lower than 150 to 200 ft. There is no evidence in the area, as far as the writer could see, to suggest that any subsidence of the valley has taken place to sink it so far below sea level. The erosive action of all rivers usually results in denuding the rocks to what is termed the "base level of erosion" which is often implicitly understood to be sea level. But, there are instances of many large rivers having eroded their bedrocks considerably below sea level for a long distance away from the sea. For example, the Mississippi has a bottom below sea level for some 400 miles above its debouchment. At several places it is said to possess a channel about 100 ft below sea level and locally even going as much as 250 ft lower.[1] According to Chamberlain and Salisbury,[2] this deep channel is the result of the erosive activity of the stream, not of subsidence. Similarly, in the case of the Godavari, the rocky bed appears to have been eroded below sea level for a distance of about 150 miles before the river joins the sea. This erosion must have established a fairly evenly "graded" rock bottom and the deltaic deposits have apparently gone on extending backwards, so much so that at the present day in this portion, the river is flowing over its own deposits, having become senile and meandering. In many places these river deposits include intercalations of very thin seams of peaty substances. Drift wood also occurs commonly in the sands. This type of wood was often cut through at various depths in some of the bore holes drilled in the area. These evidences suggest the "deltaic" character of those deposits. In this area the river has evidently passed from the stage of active erosion into one of deposition. However, above the open gorge, far higher up in the hilly tract, the "headward" erosion may still be going on.

[1] References are at the end of the paper.

In view of the fact that the rocky bed of the river has attained an evenly "graded" surface at or about 125 ft below sea level, excepting for an occasional relict island rock-outcrop or subsurface protuberance (beneath the sands), the ancient deep erosional gorge concealed by the sands and clays would surely be encountered. There is a considerable amount of evidence obtained from the geophysical survey and the bore-hole data suggesting that this deep course definitely lies all along on the right bank. In previous times, the river must have confined its course to that side but it has later shifted to a more easterly direction toward the present left bank, lowering the rock levels in that portion also, and constituting another eastern deep section of a more recent origin. These two "deeps" appear to be separated in places by a protrusive rocky medial ridge either covered up by the sands or very occasionally outcropping above as rocky islands.

I have laid stress on these geological conditions at some length because these details have an essential bearing on interpreting the geophysical indications. The problem of finding the site at which the concealed deep section has the narrowest width could only be solved by keeping the most vigilant geological control in interpreting the complicated electrical reactions.

Sites Investigated

The several sites designated *IA*, II, III, α, β, γ, and others, proposed for investigation are denoted by the respective lines marked across the river in Fig 1. Geophysical measurements were carried out at only five of these sites; while it was decided to reject the remaining ones on the strength of the data obtained in a few of the bore holes which had been put down before, or other adverse factors. In describing these sites, the term "deep section" frequently will be used to denote the portion of an alignment containing the filled-up gorge in the river bed wherein the rocks are found at levels deeper than 50 to 60 ft below mean sea level. The subsurface rock topography relative to the proposed dam alignments has been divided into a deep section and a high rock portion. In the latter, the bedrock could be met above -50 ft m.s.l. The attitude that an alignment takes in relation to the old, filled-up deep gorge is an important factor affecting the cost of the dam. From the surface features an alignment may look comparatively short, but crossing the hidden deep section obliquely proves costlier than another longer alignment which would negotiate the concealed deep section at its narrowest width. So, the various possibilities of the subsurface rock topography had to be considered in order to decide upon the most economical alignment. Plans and sections illustrating the layouts for the electrical surveys, the particulars of bore holes drilled, and the nature of the bed-rock profiles deduced, are given in Fig 2 to 8. The following notes give the summary of the results of the surveys carried out:

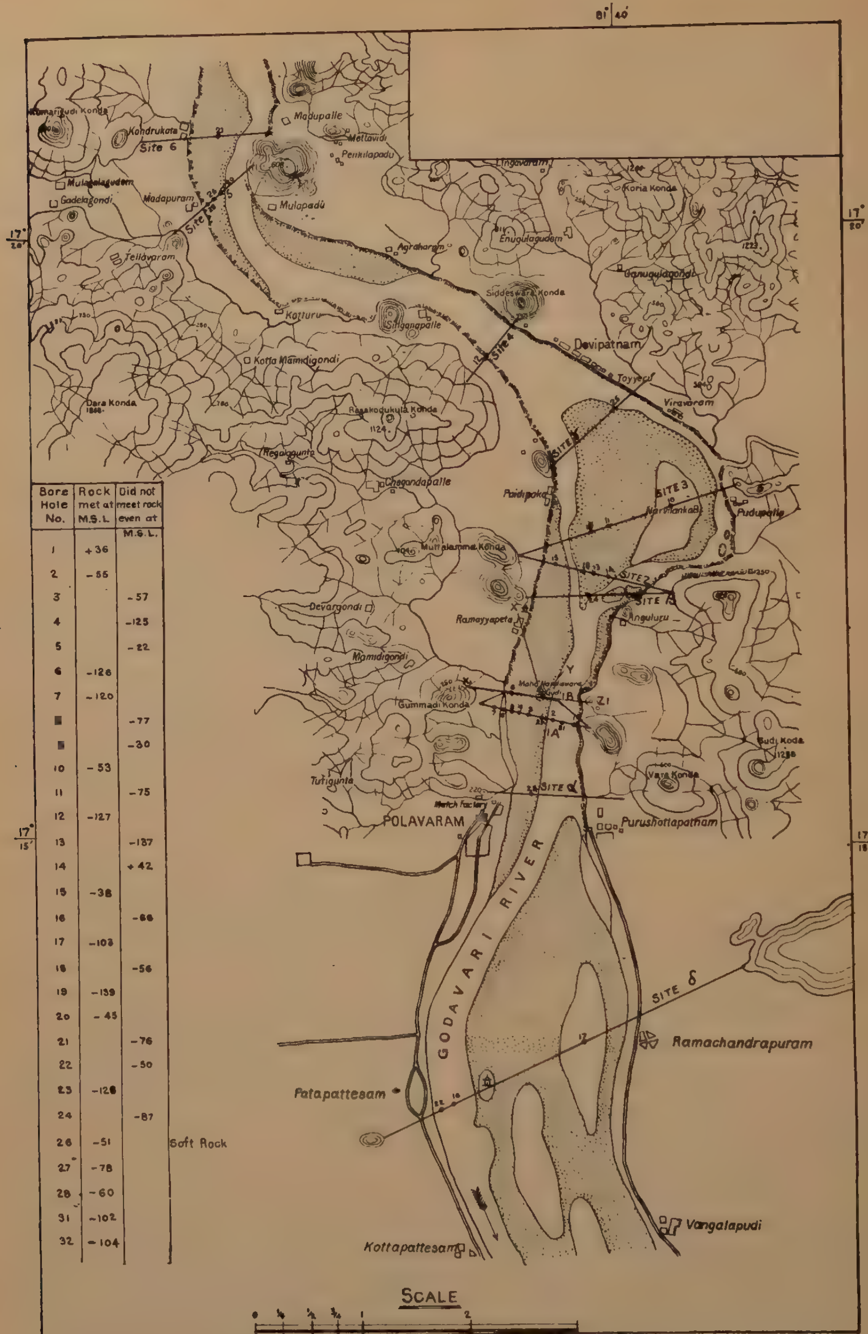
α *Site*

From the point of view of the facilities for "off take," the α site (Fig 2 and 3) is said to be the most favorable one as it happens to be situated at what may be termed as the mouth of the gorge. The alignment is 6454 ft long and on the strength of the geophysical evidence it was estimated that the deep section would be at least 5000 ft.

IA Site

This alignment is 5260-ft long and has been very closely investigated both by geophysical measurements and drilling.

Prior to the geophysical examination, seven bore holes had been put down (Fig 4). Of these, bore holes No. 1 and 2 were on the eastern and western margins of the water course and had touched rock



Bore Hole No.	Rock met at M.S.L.	Did not meet rock even at M.S.L.
1	+36	
2	-55	
3		-57
4		-125
5		-22
6	-126	
7	-120	
[illegible]		-77
[illegible]		-30
10	-53	
11		-75
12	-127	
13		-137
14		+42
15	-38	
16		-66
17	-103	
18		-56
19	-139	
20	-45	
21		-76
22		-50
23	-128	
24		-87
26	-51	
27	-78	
28	-60	
31	-102	
32	-104	

Fig 1—Map of Godavari River near Polavaram showing sites investigated.
(Extract from Topographical Map of the Survey of India-Sheet. No. 65G 11 and 12, Part.)

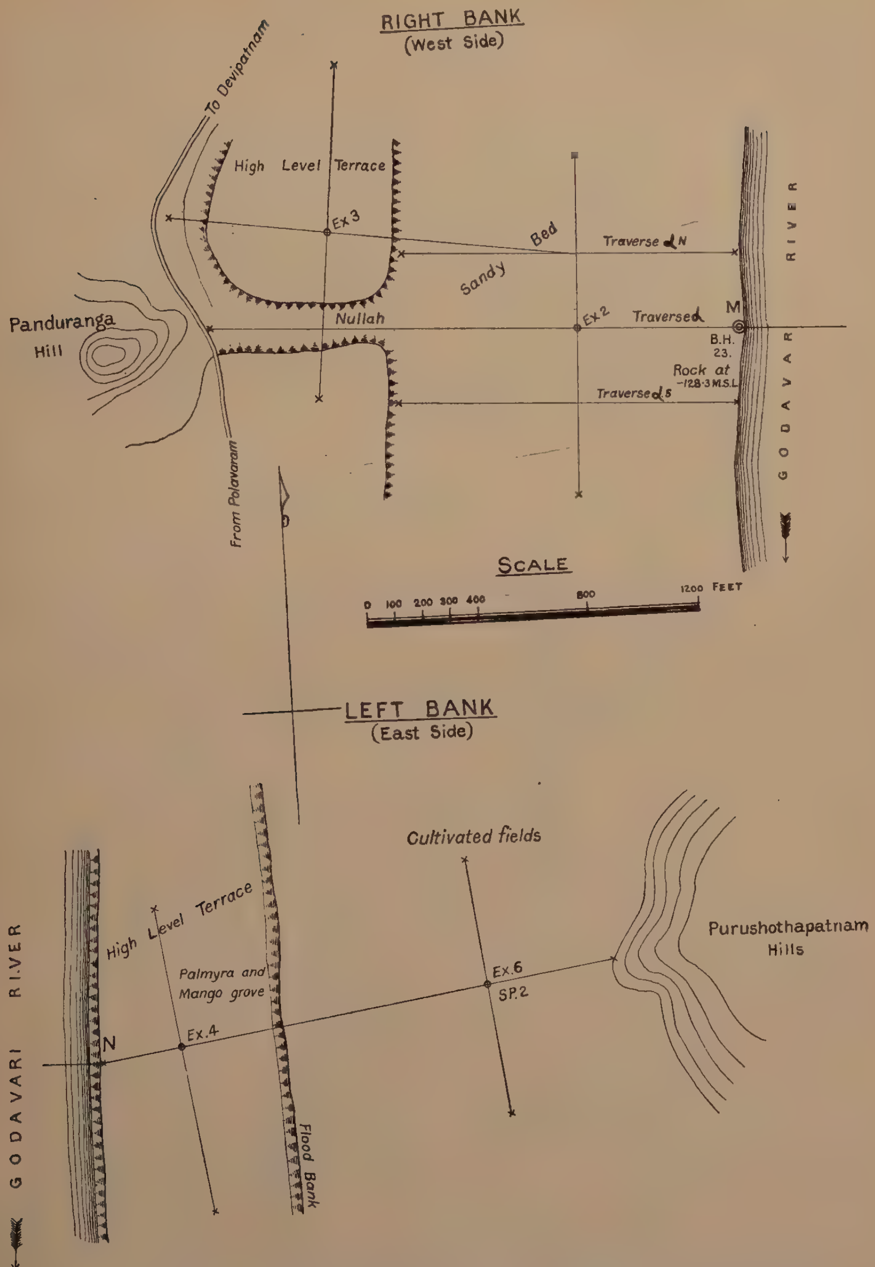


FIG 2—PLAN OF ALPHA SITE SHOWING LAYOUT OF GEOPHYSICAL SURVEY LINES.

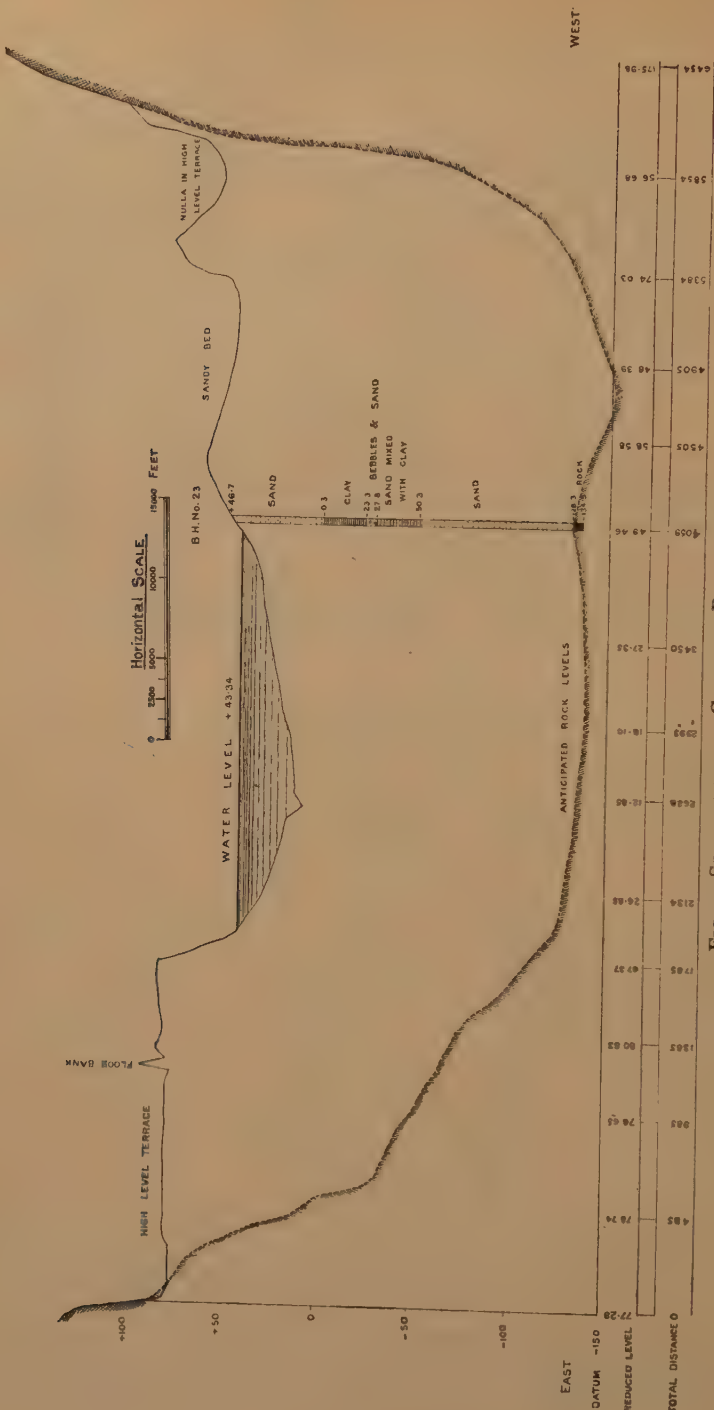


FIG 3—SECTION ACROSS GODAVARI RIVER, α SITE.

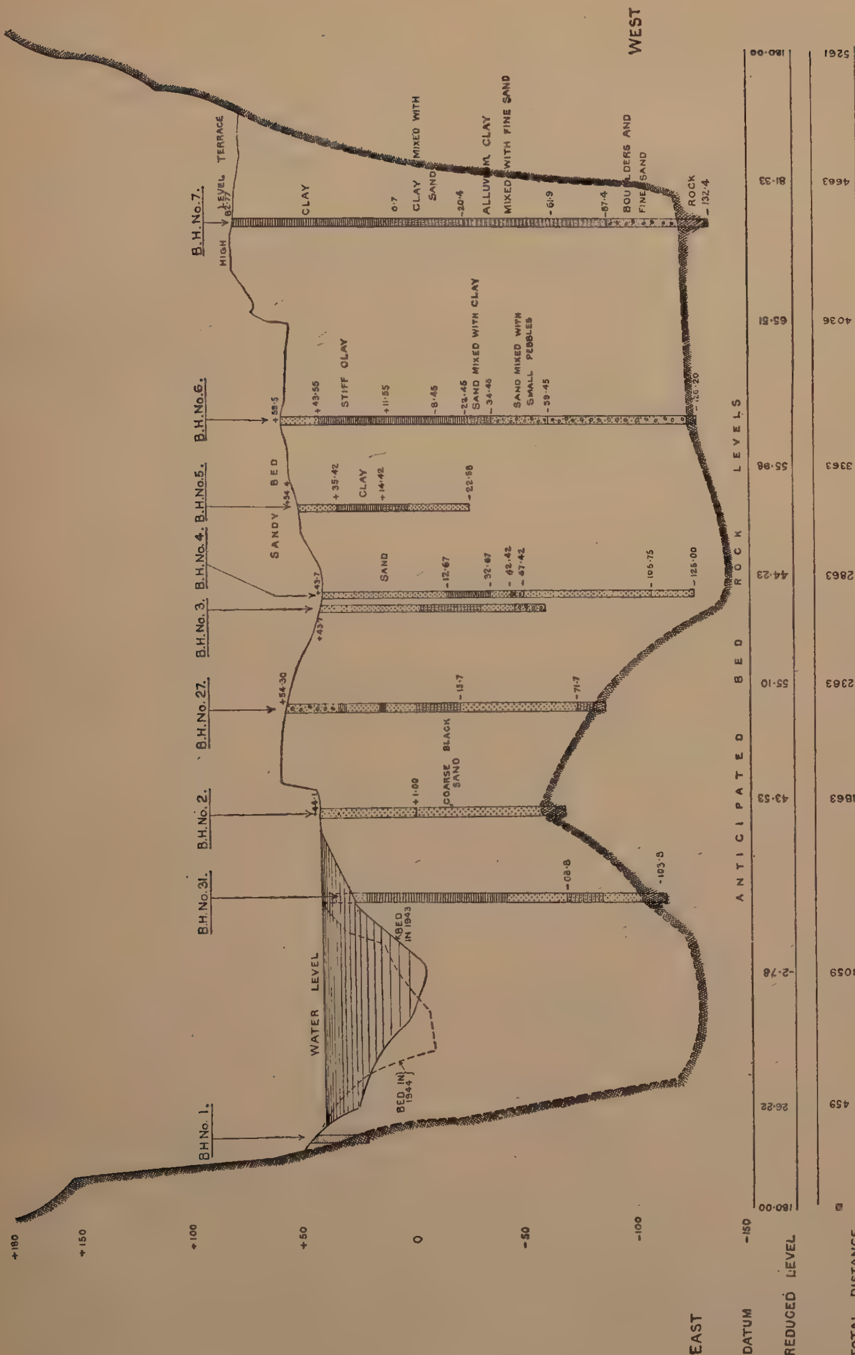


FIG 4—SECTION ACROSS GODAVARI RIVER, IA SITE.

at +36 and −55 ft m.s.l., respectively. The bore holes drilled on the high level terrace near the hill flank on the western side on the right bank had touched rock at 200 ft below the surface (about −125 ft m.s.l.) but the other bore holes put down in the sandy bed failed to meet rock even at −125 ft m.s.l. Doubts also arose as to whether the rock encountered at −55 ft m.s.l. in bore hole No. 2 was truly in situ or only a boulder.

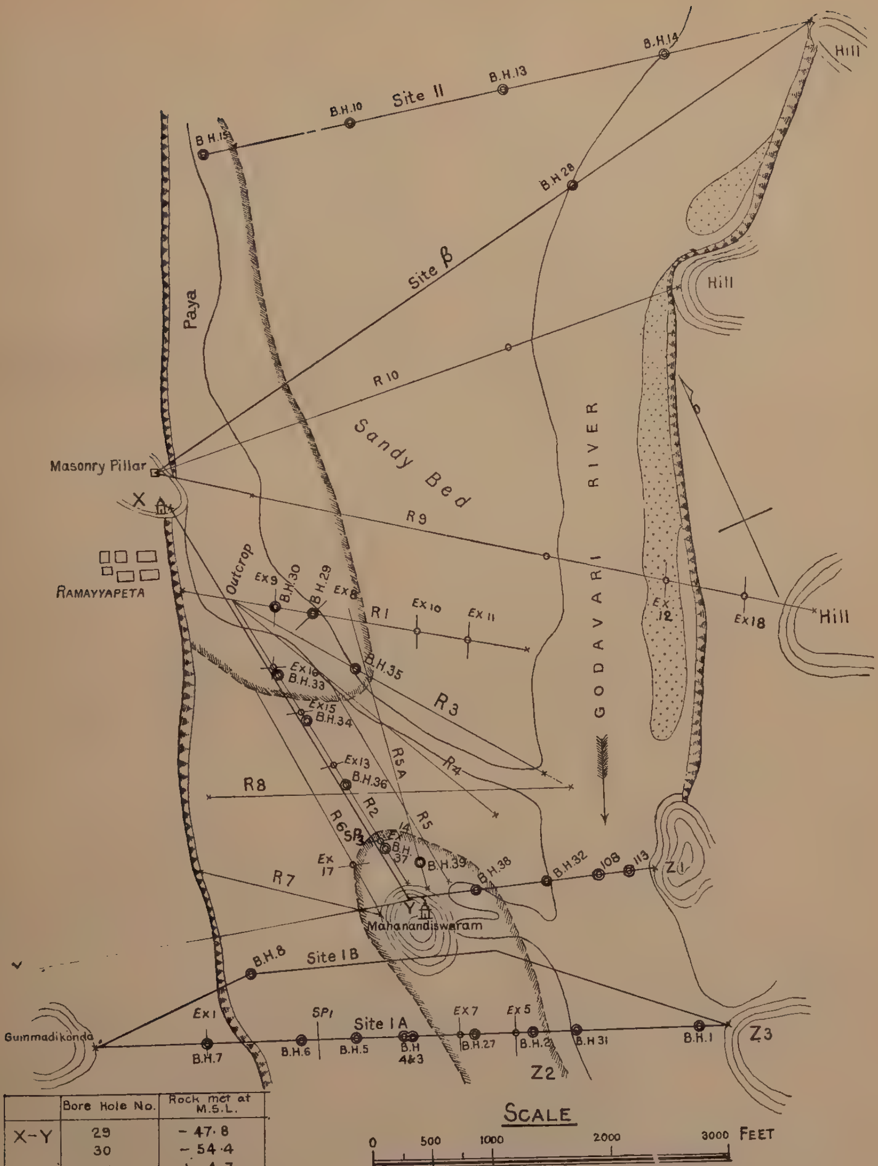
The geophysical examination at this site threw some further light on the subsurface conditions. The resistivity data indicated that within 100 yd east of bore holes No. 3 and 4 the bedrock could be met about 120 ft below the surface (or about −70 ft m.s.l.). Also, the curves obtained near bore hole No. 2 indicated the rock to be −50 to −60 ft m.s.l., confirming the "in situ" nature of the rock already touched in that bore hole. It became evident, therefore, that a high rocky protuberance was concealed beneath the sands in that part of the river bed. In order to confirm this indication bore holes No. 27 and 31 (Fig 4) were recommended to be put down and these, when completed, touched rock at −78 and −102 ft m.s.l., respectively. From the attitude of the deeper rock in hole No. 31 drilled at the water margin, it became clear that another deep section would be met beneath the present water course. Thus, from all the data collected, it was estimated that the total width of the deep section will not be less than 4400 ft or about 80 pct of the proposed alignment.

Ramayyapeta Site, Alignment X-Y-Z_1

The Ramayyapeta site originally had not been thought of for the investigations, but the geophysical surveys showed this to be the most promising. The resistivity traverses carried out in the sandy bed near Ramayyapeta indicated that the bedrock could be met within a depth of about 100 ft below the surface for a distance of over 1200 ft from the right bank. This was rather a surprising feature for in the whole of this region the deep section was known to fringe the right flank, presenting a sudden precipitous drop or slope of the underlying rock toward the river side. The resistivity data further suggested that between this hidden rocky protuberance near Ramayyapeta (X) and the MahaNandiswaram Island outcrop (Y), the "deep" intervened only as a narrow gorge.

The situation of the Ramayyapeta hill (X) is such that a number of dam alignments could be taken connecting it with one or the other of the hills on the left bank. In order to find the alignment which would permit negotiating the deep section at its shortest width, a number of radial lines R_1, R_2, R_3 and so forth, were further surveyed by resistivity methods in detail. In the accompanying plan (Fig 5) the disposition of these lines along which the resistivity profiles were obtained and the points at which the expanding electrode and single-probe tests were carried out, have been marked. From the data obtained in these detailed measurements it was possible to roughly indicate the outlines of the buried high rock portion where foundations would be necessary to a depth of only about 100 ft below the surface (or about −50 ft m.s.l.). Similarly, to the north of the MahaNandiswaram Island (Y) the rocky surface below the sand was outlined. The extent of the areas where rock may be expected within −60 ft m.s.l. has been shown in Fig 5.

Following the indications of high rock portions, the new alignment X-Y-Z_1, was proposed (Fig 5), consisting of two limbs joined at an obtuse angle at Y instead of the usual alignment straight across the river connecting the opposite flanks. In the limb X-Y which is 3656-ft long, it has been estimated that there would be about 1350 ft of deep section where the rock is −100 to −155 ft m.s.l. The resistivity profiles also indicated that the deepest



	Bore Hole No.	Rock met at M.S.L.
X-Y	29	- 47·8
	30	- 54·4
	33	+ 4·7
	34	-103·8
	35	- 86·7
	36	-156·8
	37	- 60·1
	39	- 15·3
Y-Z1	38	- 60·6
	32	-104·8
	108	-114
	113	-103

Ex = Expanding Electrode Test
SP = Single Probe
B.H. = Bore Hole
× R1, R2, R3, Etc = Resistivity Traverses
Rock Expected within -60′ M.S.L.
Dam Alignments Investigated.

FIG 5—DETAILED MAP OF RAMAYYAPETA AREA SHOWING LAYOUT OF GEOPHYSICAL SURVEY AND DAM ALIGNMENTS.

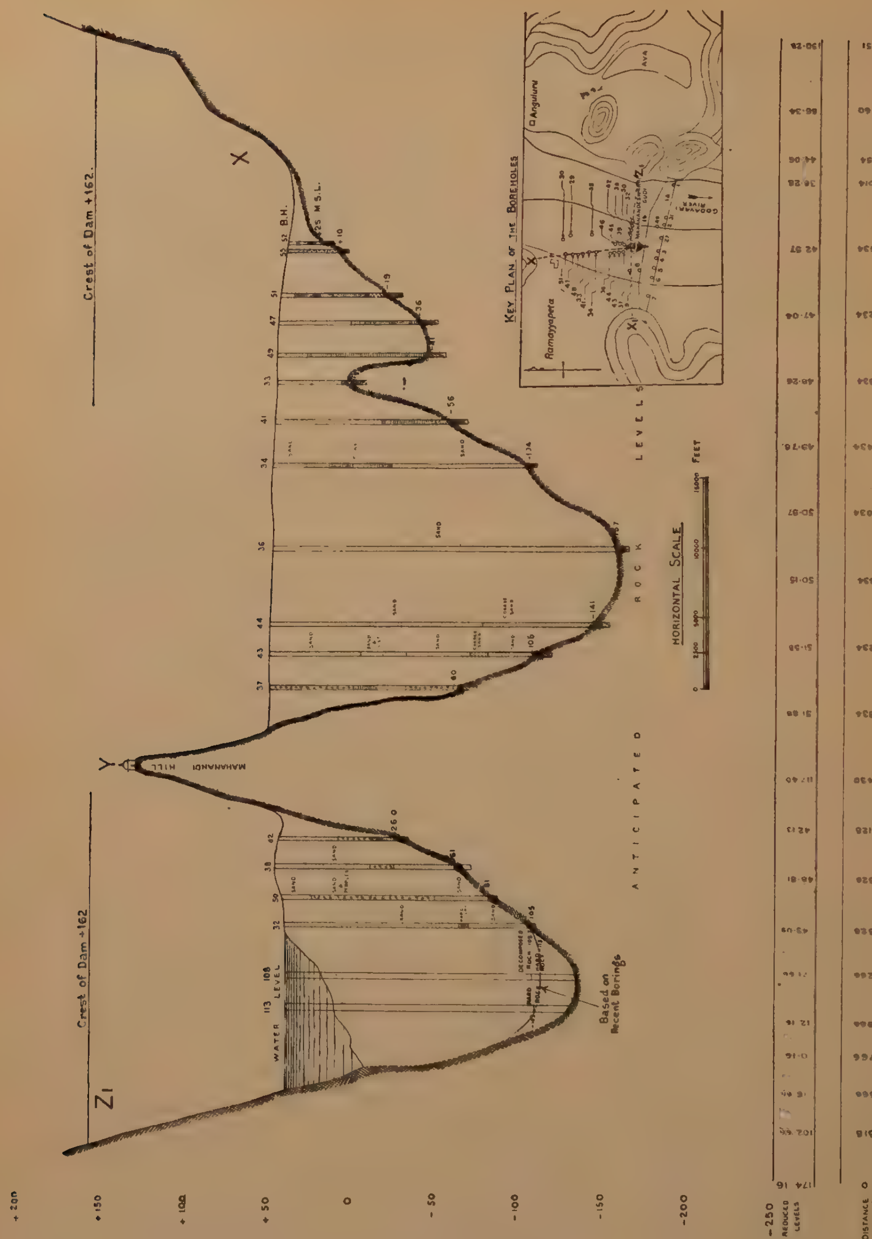


FIG 6—SECTION ACROSS GODAVARI RIVER, RAMAYYAPETA-MAHANANDISWARAM ALIGNMENT X-Y-Z_1.

portions (−150 ft m.s.l. or lower) may be limited to about 250 to 300 ft on this alignment.

The other limb Y-Z_1, much of which is covered by water, is about 1680-ft long, and some attempts were made to ascertain the level to bedrock by resistivity methods over the water course. This indicated another deep section below the water course where the rock level may be as low as −120 to −140 ft m.s.l. Probably, a length of about 1200 ft of the limb Y-Z_1 should be regarded as having the deep section.

Taking the whole of the X-Y-Z_1 alignment into consideration, the total extent of the "deep" amounts to 2900 to 3000 ft, or in other words, this site will have some 50 pct of its alignment as high-rock portions. A few bore holes were put down at the site to verify the electrical indications and the results were satisfactory, checking the indications very closely in many cases. This verification having established the merits of this site, its selection was recommended.

Madapuram and Kondrukota Sites, Alignments V and VI

Although from the very outset it had been realized that the "off take" from these uppermost sites would be exceedingly difficult and costly, they were examined in detail (Fig 7) to test the possibilities of discovering rocks at high levels for the dam, in case there were any ancient water falls buried in that region.

The resistivity curves obtained at these sites did not permit a satisfactory solution, because of the complicating features introduced by the occurrence of thick clay beds mixed promiscuously with sandy and pebbly layers. Moreover, the bedrock in this area is a highly conductive medium, probably because of extensive decomposition. However, the curves were interpreted taking the geological probabilities into consideration and it was possible to draw a tentative boundary line demarcating the extent of the deep section, which no doubt lies concealed here. A small protrusive medial rocky ridge (Fig 8) buried beneath the sands in the river bed was found to be constituted of highly weathered rock below the sands. Such decomposed rock would have to be excavated completely down to hard, fresh rock for securing the foundations of the dam so the apparently promising high-rock levels noted in a portion of this site offered no saving of cost.

From the investigations carried out, it was clear that the deep section of at least 2500 ft on alignment V and about 3000 ft on VI had to be encountered. If alignment V whose total length is only 4860 ft (including 2500 ft of the deep section) were chosen, there might be some economy in the foundation costs as compared with any of the lower sites. But against this, the disadvantages for the "off takes" involving tunnelling in hard rock for a few miles at least, outweigh the meager economy that could be secured in the dam foundations. Both alignments V and VI were finally recommended to be ruled out of consideration for the project.

Final Selection

From the foregoing account, it will be obvious that no other site in the area offers the same favorable conditions as the Ramayyapeta-MahaNandiswaram site. Consequently, it had been recommended for more intensive examination by bore holes to work out the costs. Subsequent to the geophysical surveys, much progress has been made in this direction, and from the information kindly furnished by the chief engineer in charge of the project, it is understood that a modification of the alignment is under consideration. According to this modification (X_1-Y-Z_1), the two limbs of the dam line, instead of forming an obtuse angle at Y, will run straight across the river. No geophysical surveys

Bore Hole No.	Rock Met At -M.S.L
20	-44·7
26	-60·2
28	-59·1
21	No Rock Even at -81·6

FIG 7—DETAILED MAP OF SITES V AND VI SHOWING LAYOUT OF GEOPHYSICAL SURVEY LINES.

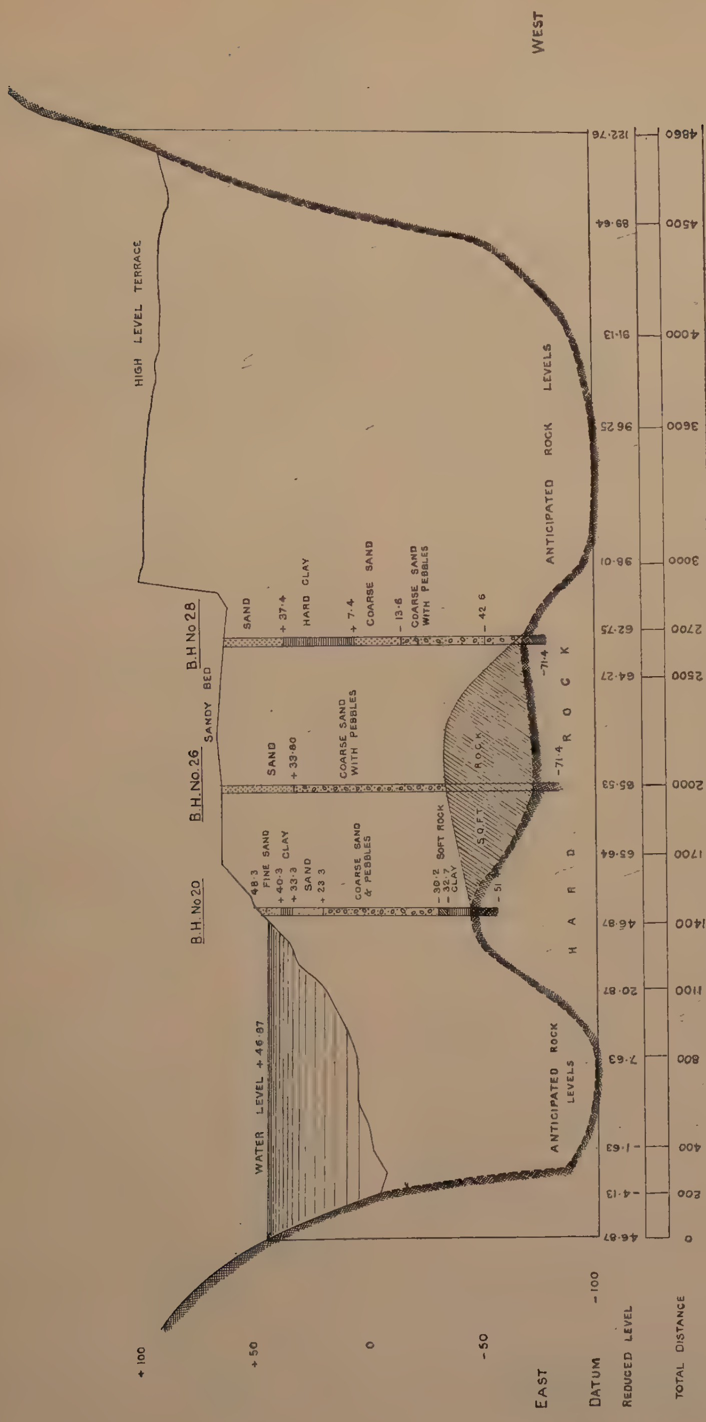


FIG 8—SECTION ACROSS GODAVARI RIVER, MADAPURAM, SITE V.

had been done to the west of MahaNandiswaram Island, but from the investigations carried out at $1A$ site, it was known a deep section would be definitely met over almost the entire width of the river bed between the MahaNandiswaram Island outcrop (Y) and the right flank hill (X_1). However, it would now appear from the point of view of relative costs which are being worked out, that the shorter alignment X_1-Y is more promising, although it negotiates a slightly greater width of the deep section as compared with the much longer X-Y line. Although the final alignment for the dam has not yet been definitely settled, the modified line X_1-Y-Z_1 is tentatively fixed as it is the shorter and more economical one.

Resistivity Methods Adopted and Apparatus Used

Having given an idea of the geophysical investigations carried out to select a suitable site for the dam, I shall now present a critical review of the experience gained, of the methods adopted, the apparatus used and the interpretative technique employed—illustrating my remarks by a reference to some of the curves obtained during the investigations.

The following systems of resistivity measurements were adopted:

1. *Expanding Electrode Test.*—The electrode-separations conformed to the usual Wenner[3] electrode configuration. "Lee-partitioning"[4] was adopted in some cases. However the measurements in most cases were taken by expansions along two mutually perpendicular lines. Generally speaking, in these expanding electrode tests and the "Lee-partition" curves the two sets of curves possess a remarkably clear agreement in their relative shapes, although in so far as the absolute values of the specific resistances (ρ_a) are concerned, there have been very large differences in a number of instances. For the interpretation of the curves, however, the relative shapes and the characteristic gradients are more essential than the absolute values of the resistivity; so the departure observed in these ρ_a values did not detract from the usefulness of the data.

2. *Single-probe Test.*—Only three cases were investigated using this electrode layout and the results obtained were rather unsatisfactory. Several factors limiting the application of this method were found in the area. Considerable irregularity in the distribution of the potentials was observed in the neighbourhood of the Home electrode, and even within comparatively short distances the earth resistance between the potential spikes would drop off to a small fraction of an ohm, which was difficult to measure accurately with the Megger Earth Tester we used. The three single-probe tests in the area were carried out for comparison and experimental purposes.

3. *Resistivity Traverses.*—To obtain profiles of the resistivity values along particular lines, traverses were carried out with the Wenner configuration of the electrodes and adopting multi-separations of 80 ft, 120 ft, 160 ft, 200 ft and so forth, repeated along the lines. This procedure has helped to outline the anomalies, often enabling us to get an approximate idea of the subsurface rock levels, or to "follow up" the anomalies, which were connected with the known features elsewhere, into unknown ground.

4. *Apparatus Used.*—The resistivity measurements were made by using (a) the Megger Earth Resistance Tester, and (b) the dc potentiometer with H.T. batteries, milliammeter, and a pair of non-polarizing electrodes.

The Megger Earth Resistance Tester, made by Messrs. Evershed and VigNoles, Ltd., London, had, as usual, four ranges 0–3, 0–30, 0–300 and 0–3000 ohms. In areas covered by clays, as in the high level terraces, the measurable resistance was often less than 0.1 ohm. This instrument

in such cases was of little use and the dc potentiometer, H.T. batteries, milliammeter and non-polarizing electrodes were used.

The potentiometer used was a design of Mr. A. Broughton Edge and manufactured by Messrs. H. Tinsley & Co., Ltd., London. This instrument had two potentiometer circuits, one main and the other auxiliary (for balancing the stray currents and natural emf in the ground) with a Weston Galvanometer incorporated in the same panel. The non-polarizing electrodes were of a new type, the pots being constructed with silver rods charged with silver chloride and immersed in 10 pct solution of pure sodium chloride.

For each measurement of the earth resistance voltages from the H.T. batteries were applied to the ground using two steel current electrodes for the purpose. The current was measured by the milliammeter and the p.d. picked up by the non-polarizing electrodes was balanced out by the main potentiometer. Before applying the current into the ground, the natural earth potentials were compensated by the auxiliary potentiometer circuit, and for each electrode separation two or three different amperages were put in, the current being reversed through the ground each time. Thus, four sets of readings were obtained and the mean resistances were accurate up to ±0.005 ohm.

Comparative tests, using both the Megger Earth Tester and the dc potentiometer at the same spot, were carried out in a few places. The discrepancies noted between the two sets of values were quite small.

Interpretation of the Curves

The interpretation of the earth-resistivity curves which were obtained by the foregoing systems of measurements, was based on the three well recognized methods for solving the depths to various layers: the empirical rule, Tagg's method and the logarithmic method of Irwin Roman. It was impossible under the conditions observed in the area to stick to any one general rule or method, and the interpretation had to be based on experience and geological probabilities.

1. The empirical rule[5] which states that "if at a certain depth there is an abrupt change of resistivity due to the presence of a body of much higher or lower resistivity than the surrounding medium, it would produce an abrupt change in the apparent resistivity electrode separation curve, at an electrode separation equal to the depth of geological discontinuity," cannot be applied, mechanically, to locate all the variations in a series of stratified beds. Nevertheless, there are numerous instances in which this rule has worked very well. As pointed out by Tagg,[6] in certain exceptional cases, when the disturbing bed is thin and is of a resistivity differing considerably from its surroundings, this empirical rule can be expected to give satisfactory results. In the area under report, the bedrock was usually a little decomposed for a few feet just below the sands. This thin layer of decomposed rock is highly conductive and sometimes caused a characteristic effect on the resistivity curves, which was easy to recognize. It has been described in this report as a "key indication" and has been used in the interpretation of the curves. At times, however, when thick clay beds intervened or when the rock itself was decomposed to an appreciable depth, this peculiar indication could not be easily recognized.

2. Tagg's method[7] was applied to most of the two-layer cases in the area but when the method was extended to the three-layer cases, of which there was a larger number, the solution of the depths was more uncertain. The determination of the value of ρ_1[a] or the resistivity of the

[a] The notations ρ_a, ρ_1, h, a, k, and so forth, used in this paper have the same significance as mentioned in Tagg's article.[7]

surface medium introduced complications and uncertainty in using this method.

Especially in the sandy bed of the river the surface was quite dry but within a few feet abundant moisture was present which gradually increased with depth until the saturation zone was touched. We thus had an upper medium which, instead of possessing a uniform value of resistivity, had variations both in the lateral and vertical dimensions. Because of this zone of saturation, most of the curves obtained in the sandy bed turned out to be three-layer cases even though the formations were only two in number—the sandy layers and the bedrock. Sometimes the top medium was quite thin and the curves could then be treated as two-layer cases, but even then a reasonably accurate indication of the depth to the bedrock could not be obtained.

3. The logarithmic method of Irwin Roman[8] was found to present a distinct advantage over Tagg's method because the value of surface resistivity, which was difficult to measure accurately, could be obtained directly from the fittings made on the theoretical curves. Because the laborious computations and plotting required in Tagg's method were unnecessary in the logarithmic method, we were able to apply the latter to almost all the resistivity curves obtained in the area. The results, however, were not always satisfactory. Many cases failed to give an unequivocal solution and the best fitting was a matter of doubt. This trouble arose more especially when the k values approached 1. In some of these cases the gradients in the apparent resistivity curves became so steep that unique fittings were exceedingly difficult.

In our investigations, all the three methods were used in suitable cases. As between the methods themselves, the results were found to vary a good deal. Had not the geological probabilities been taken into consideration, the estimation of the depth to bedrock would have been misleading in a number of the cases so interpreted. The bore holes put down to verify the indications have demonstrated, in several instances, the value of empirical method and very often by following up this key indication in the curves a satisfactory estimation was made of the depths straight from the plot of the apparent-resistivity-electrode separation curves. As previously noted, just beneath the alluvium the bedrock is usually decomposed for 2 or 3 ft and this forms a highly conductive layer which causes a characteristic indication at an electrode separation closely corresponding to the depth of this thin conductive layer. This characteristic indication formed a "key," as it were, to the interpretation of the curves and often we had to be guided by such an empirical procedure, especially when the other methods yielded improbable values, or the curves were not at all amenable for any rational analysis.

The various resistivity curves obtained during the investigations could be grouped into five general types depending upon their general shapes. A few selected cases to illustrate each of these groups are explained as follows:

Type 1.—These curves are among those obtained on the high level terraces where a thick clay bed forms the surface layer which usually has a very low resistivity. Except for the minor inflection of the curves in their upper portions, which may be attributed to the water table, the resistivity gradients generally rise very high. For all practical purposes, they may be treated as only two-layer cases with an insulating bed below the clays.

In Fig 9 the two curves are divergent up to 80-ft separation but are coincident for the larger separations. The empirical indication is at about 80 ft. The application of the logarithmic method gave 90 ft as the depth to lower resistive layer. The bore hole (B.H. No. 7), near by, had shown that

the clay was about 85 ft thick and still lower sands and pebbles prevailed up to a depth of 203 ft, below which hard bedrock was found. The depth to the resistive bed deduced from such curves related only to the sands below the clays. There was no indication whatever corresponding to the bedrock on the curves. Probably, there was no definite electrical discontinuity corresponding to the interface between the sands and bedrock in these cases. When a highly conductive bed overlies two successively resistive beds, the indication will pertain only to the first interface in contact with the lower portion of the top conductive bed. If, however, there were a second conductive layer separating the two resistive beds, another indication might have been noticeable at a larger electrode separation, but in the present instance the two resistive beds, namely, the sands and the bedrock, are apparently electrically continuous and indistinguishable.[a]

Type 2.—The first part of the curves of this group shows high resistivity values that fall off rapidly with a very sharp gradient. The ρ_a values become a minimum at some separation and then rise with a fairly steep gradient for the further increased separations. This is quite a typical effect observed in three-layer cases where a conductive layer is interposed between two resistive beds. In the majority of the instances, the inflection appears to be purely a functional characteristic depending upon the values of h_1 and h_2, and also of k_1 and k_2, and the like, of the beds involved. Even when such inflection directly corresponds to the depth of the lower

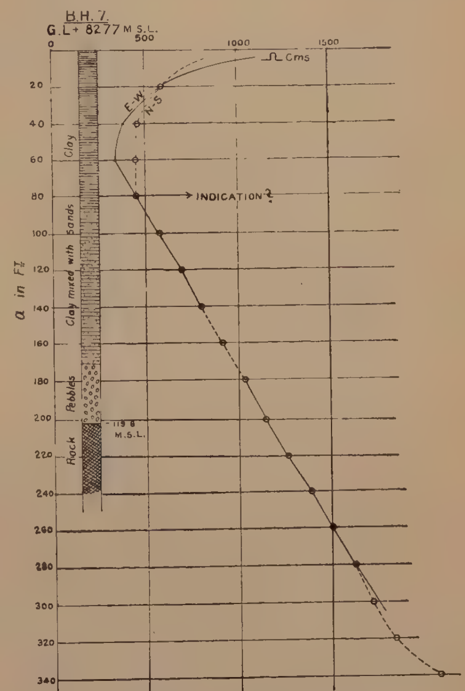


FIG 9—EXPANDING ELECTRODE TEST No. 1, HIGH LEVEL TERRACE, 1A SITE.

resistive bed, the indication may not be caused by bedrock but only by a second layer of sands beneath the clays. However, in a good many instances in this area, the bedrock beneath the sands (or pebbly layers) is decomposed to a very small depth, about 2 or 3 ft, and has become highly conductive. This usually causes another significant indication which can be recognized on carefully scanning the anomalies. When once this indication is established by correlation with a bore-hole evidence, it could be used as a key in interpreting other curves of a similar nature in the area.

For instance, in Figs 10 and 11, the geological probabilities were against inter-

[a] The writer has since tried the cumulative-resistivity plotting method of R. Woodward Moore[9] to some of the curves dealt with in this paper. The Moore method in some cases has definitely disclosed indications of the bedrock depths (as well as the depths to the sands beneath the clays) in close agreement with the actual depths recorded in the respective bore holes. Undoubtedly, Moore's empirical method also forms a valuable means of successfully attacking the apparent resistivity curves which fail to respond to the Gish-Rooney, the Tagg or the Roman logarithmic methods. The writer is making an extensive study of both the cumulative and differential plotting methods of a number of resistivity curves he has already investigated and hopes to communicate the results in the form of a separate paper.

preting the first inflection of the curves, or the values of h_1 and h_2 worked out by the Tagg and Roman methods, as attributable to the depth of bedrock. By a careful

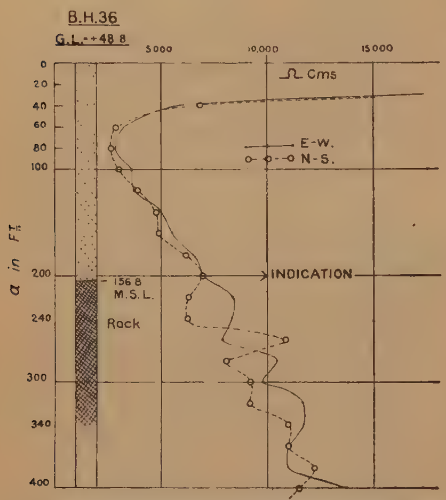


FIG 10—EXPANDING ELECTRODE TEST No. 13. SANDY BED, RAMAYYAPETA SITE.

study of the anomalies in the curves, it was possible to detect a key point in the lower rising resistive gradient portions, which suggested merely some electrical discontinuity. In Fig 10, it may be noted that both the east-west and north-south curves follow closely up to a certain stage and then show a rather simmering fluctuation just after 200-ft separation is passed, and the two curves diverge with numerous other minor breaks for the still larger separations. The bedrock was estimated to lie at this depth. Bore hole No. 36 drilled later at the spot, touched the bedrock at a depth of 205.6 ft, thus confirming this electrical indication. Although in the bore-hole log furnished a clay bed has not been recorded to explain the upper conductive layer causing the inflection of the curve at 60 to 80-ft separation, it is known from the evidence of the other bore holes that a small clay bed must have been encountered at some depth of the surface sands, this minute detail having probably escaped the notice of the drilling staff.

In Fig 11, the three curves are not in very good agreement until 140-ft separa-

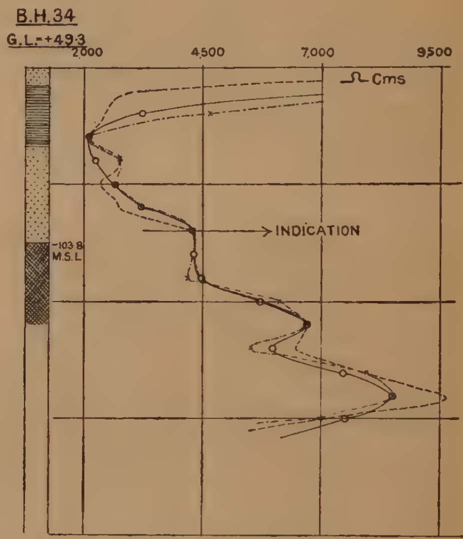


FIG 11—EXPANDING ELECTRODE TEST No. 15. SANDY BED, RAMAYYAPETA SITE.

tion, but become concordant on the still larger separations. The indication at 140-ft separation by an abrupt drop in the resistivity gradient was taken as a key for estimating the depth to bedrock. The logarithmic method gave 51 ft for h_1 and for the lower resistive bed (h_2), the fittings were rather ambiguous giving values of 96, with 100 and 135 ft as the depth. Since the empirical indication suggested a depth of 140 ft, the bedrock was "announced" to be met at about that level. Bore hole No. 34 drilled at the spot, however, touched rock at 153 ft. A thick clay bed was met up to 60 ft in depth to explain the top conductive layer, and with regard to the bedrock it was definitely found that highly decomposed gneiss was touched at a depth of 150 ft and the hard, fresh rock below 153 ft.

Type 3.—This may be taken as a special variety of the foregoing group, in the sense

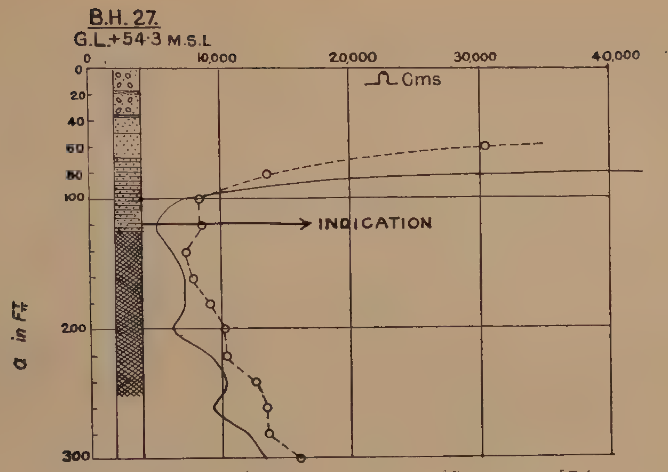


FIG 12—EXPANDING ELECTRODE TEST NO. 7. SANDY BED, IA SITE.

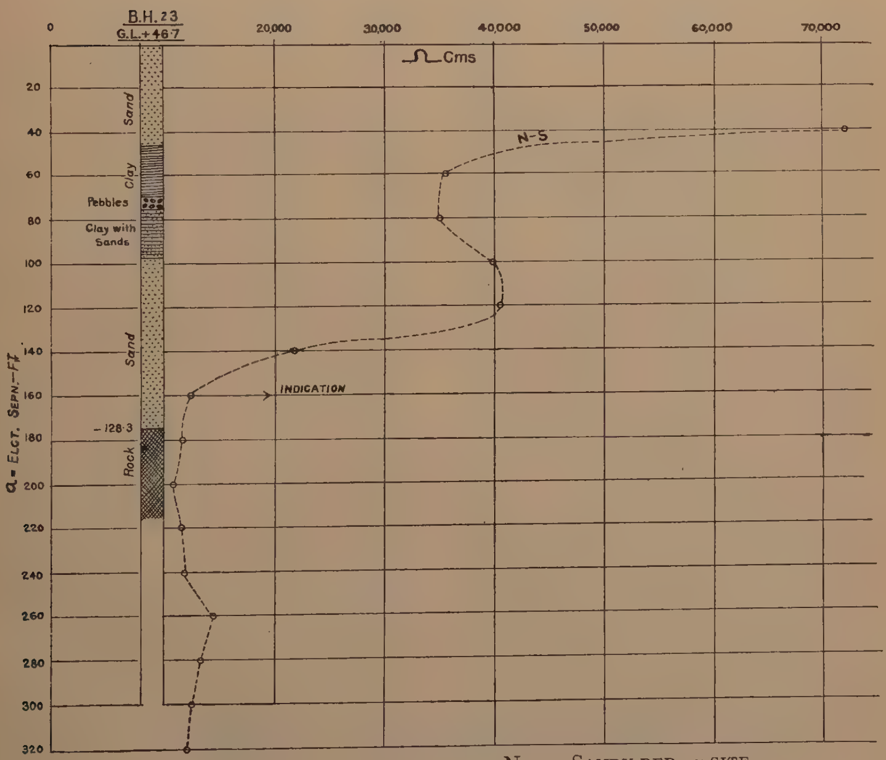


FIG 13—EXPANDING ELECTRODE TEST NO. 2. SANDY BED, α SITE.

that the rise of the resistive gradient beyond the inflection points of the curves is far less marked. The key indication was clearly noticeable in this type also. Referring to

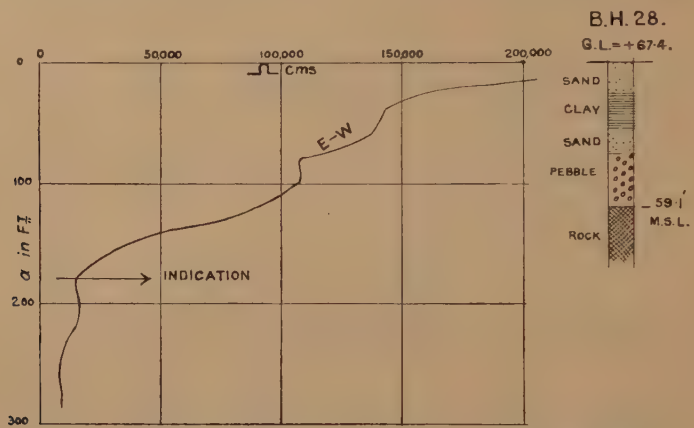


Fig 14—Expanding electrode test No. 21. Sandy bed, site V.

Fig 12, it may be stated that the surface resistivity amounted to 175,000 ohm-cm (omitted in the plotting) but dropped off to only 7500 ohm-cm at 100-ft separation. That this extremely large drop is caused by variation vertically, and not by any surface or lateral irregularity, is established by the fact that the curves on both the north-south and east-west expansions have yielded a good agreement on this anomaly. For separations larger than 100 ft, the ρ_a values vary very little but nevertheless show a small steady gradient (if the minute fluctuations are disregarded). The key indication was placed at 120 ft depth. For application of the logarithmic method, the upper part of the curve was found to have too steep a gradient to obtain a proper fitting for h_1 layer, but the fitting for the lower part of the resistive curve (h_2) gave 132 ft. Actually, bore hole No. 27 put down here met kaolinized rock at a depth of 129 ft and the harder gneissic rock at 132.5 ft.

Type 4.—The curves of this type showed high surface resistivity which dropped off to very low values at some large separation and then remained more or less constant with little or no rise for further expansion of the electrodes, thereby denoting the existence of a lower conductive bed instead of the usual resistive bed. In these cases, the bedrock is probably decomposed to some extent and behaves as a conductor compared to the sands and other unconsolidated formations overlying it. It may be that the decomposition zone does not form a thin, "finite" layer possessing crisp boundaries in this case and thus the curves lack the clear definition of a typical three-layer case where the lower resistive bed also comes into the picture after the inflection.

In fact, if one were to notice especially the gradual evolution of the curves from Type 1 to 4, it will be obvious we are dealing with rather a varied condition of the bedrock; evidently there must be decomposition to a varying extent at different locations.

In Fig 13 it may be noted that the high ρ_a values fall off to about 35,000 ohm-cm at 60 to 80-ft separation, show a slight rise after this, and again fall off rapidly after 120 ft becoming steady or almost constant beyond 160-ft separation, which was regarded as the key indication. Bore hole No. 23 drilled close by met rock at a depth of 175 ft. The bore-hole log has also re-

corded a clay layer at 60-ft depth corresponding to the first conductive anomaly on the curve.

Fig 14 shows the curve obtained at a different site. In this case the key indication was taken at 180 ft, but bore hole No. 28 near it, touched rock at a depth of 126.5 ft, putting the interpretation in considerable error. These curves were found even more unsuitable for application of the Tagg's method and the logarithmic fittings.

Type 5.—This type was very rarely found in the area and had altogether a different shape, showing a question mark (?) shape when compared to the foregoing types. The intermediate layer in this type is a sandy bed, more resistive when compared with the surface-clay layer and the bottom bedrock. Because of an appreciable decomposition in bedrock, there is again an electrical discontinuity below the sand. The key indication in the curves of Fig 15 was taken at 160-ft separation. No bore hole was recommended to be drilled at this spot to test the indication, but the existence of a deep section here with rock as low as 180 to 190 ft below the surface has since been established by drilling on an upper alignment.

Resistivity Profiles

The foregoing interpretative aspects pertain to the expanding electrode and single-probe curves. I shall now briefly touch upon the interpretation of the resistivity profiles obtained by the multi-separation traverses along the various lines. Here again, the interpretation had to be largely empirical and consisted mostly of following up anomalies correlated with a known configuration in one part into an unknown ground. The resistivity profiles furnished indications of the lateral variations coupled with vertical (depth) variations. The anomalies depended upon (1) the variations of materials, laterally as well as vertically; (2) the respective thickness of the materials; and (3) the surface irregularities. The last factor, however, could easily be appreciated and taken care of by a direct observation of the variations in ground surface along the line of measurements. With regard to the first two factors men-

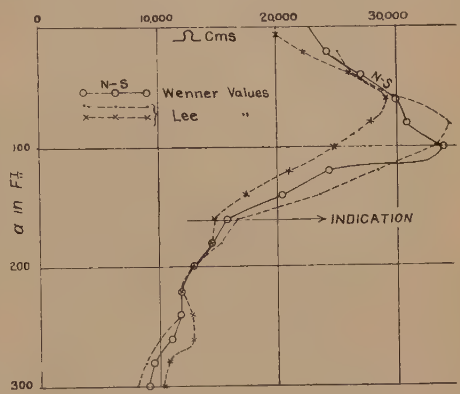


Fig 15—Expanding electrode test No. 22. Sandy bed, site V.

tioned, the appreciation of the curves had to be done by taking into account a combination of both the lateral and the vertical anomalies. The breaks in the chain of the profile-curve at once revealed the abrupt changes in the character of the materials laterally. In case there was merely a change of the subsurface levels of the bedrock lying beneath the overburden, these changes were "reflected" in the increased electrode separation values. For instance, a key indication corresponding to the depth to bedrock would occur on the corresponding electrode separation over the "affected" stations, denoting the variation of the levels. The profile data were viewed in a criss-cross way, as a series of expanding electrode curves joined together successively at each of the stations on the traverse line. Usually, however, there were variations both in the nature of the alluvial materials and the bedrock levels over long traverses across the river bed. In such cases the anomalies became more complicated and the interpretation had to be based on analogies and comparisons from previous experience gained by studying the bore-hole results in different parts.

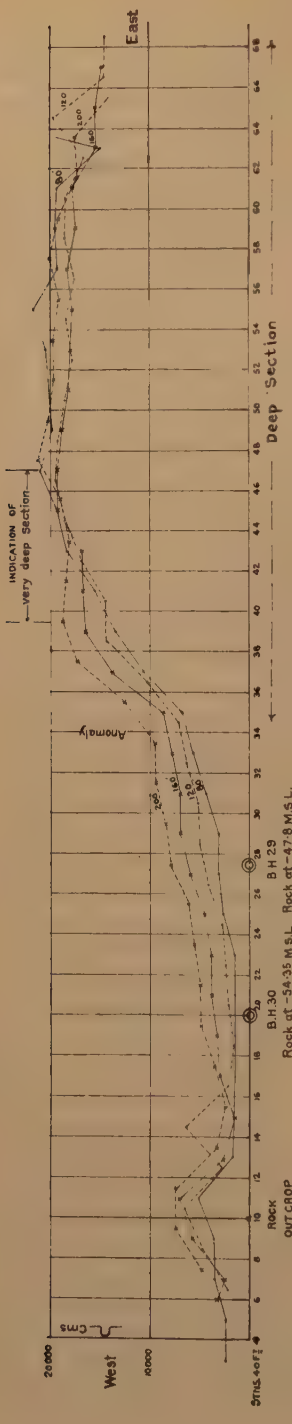


Fig 16—Resistivity profiles R_1, Ramayyapeta site.

Supplemental expanding electrode tests at a few selected points on the traverse furnished additional data, sometimes aiding in the interpretation of the profiles. In any event, these resistivity profiles could only be used to get a qualitative idea of the subsurface conditions, judging the depths to bedrock only in a relative manner. The anomalies pointed to the "shallows" and "deeps" in a general way and made it possible to follow up these indications in adjacent areas.

As examples of the resistivity profiles Fig 16 and 17 may be noted. In R_1 (Fig 16) the anomaly at station 34 is of a striking character. Although on the surface the ground over which the traverse was run was covered by sands and showed apparently no variations whatever, the resistivity profiles revealed an interesting feature in the subsurface rock topography. From 0 to 34th station, the sands are underlaid by clays which in turn are intercalated with sands or are overlying the bedrock at fairly high levels. In the more easterly portions beyond station 38 the sandy layers are free from the intercalations of clay. The successive rise of the ρ_a values with increasing electrode separations is a consistent feature up to about station 40, but further on the curves tend to club themselves together. The anomaly about station 34, indeed, denotes a transition in the subsurface rock levels, the eastern portions going down into a deep section whereas the western portion has rock at fairly high levels. The resistive peak at station 10 is actually the result of a small outcrop of rock. Attention may also be drawn here to the small "trough" in the profiles between stations 40 to 46. In no other part of the profiles do the curves hug each other so closely, and this anomaly shows up the deepest section of bedrock, a narrow deep creek with rock levels as low as −150 to −180 ft m.s.l. within the wider, deep section. The occurrence of such a narrow, deeply eroded channel in the bedrock has

been indicated in many other alignments in the area.

R_3 profiles in Fig 17 have a feature similar to that of R_1 (Fig 16). The indication of the deep section beyond the anomaly point in the curves at station 15, and also of the more localized narrow, deep creek between stations 18 and 22, is noteworthy and illustrates how sometimes the features may be followed up clearly from one traverse to another.

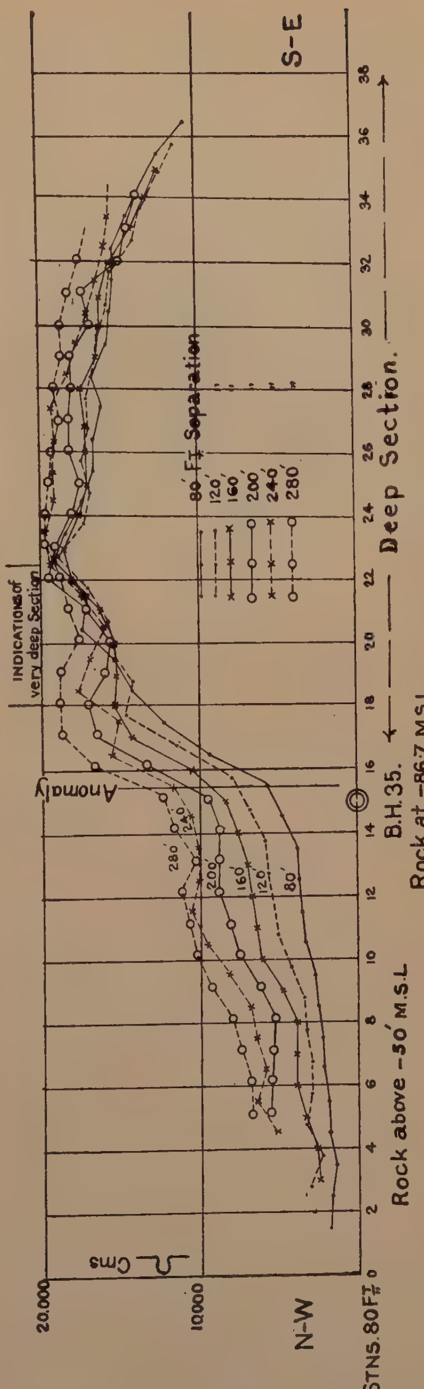


FIG 17—RESISTIVITY PROFILES R_3, RAMAYYAPETA SITE.

RESISTIVITY MEASUREMENTS OVER THE WATER COURSE

The resistivity determinations over the water-covered portions of the river were wholly of an experimental nature. This was the last to be taken up in our geophysical investigation in the area. It was reserved to be tried only at the place where the investigations on the land had already enabled the selection of a suitable alignment for the dam. With the exception of two of the lowest sites the water course did not constitute in that season a width of more than 20 pct of the proposed alignments. The conditions in such a small part of the site could affect the merits but little, and so the geophysical investigations of this part was left until the other land portions were fully investigated.

The geophysical surveys and the borehole investigations had shown that the Ramayyapeta-MahaNandiswaram site was about the best possible in the area, therefore, it was decided to extend the resistivity surveys into the water-covered portions to the east of MahaNandiswaram Island in order to obtain an idea of the depth to bedrock.

In the water course between Y and Z_1, (Fig 5 and 6) one expanding electrode test was carried out. The width of water here is about 900 ft and its depth about 50 ft in the deepest part which is in the middle portion. Three boats were employed. One of them, with the Megger earth tester and the operator, was stationed in the middle

of the river by anchoring. The other two boats were kept mobile and served to maintain the required line of measurements. The electrode contacts in water were made obtaining the true resistivity of the water. The measured resistances were uniform throughout, varying between 3.4 to 3.5 ohms. The resistivity of the water was

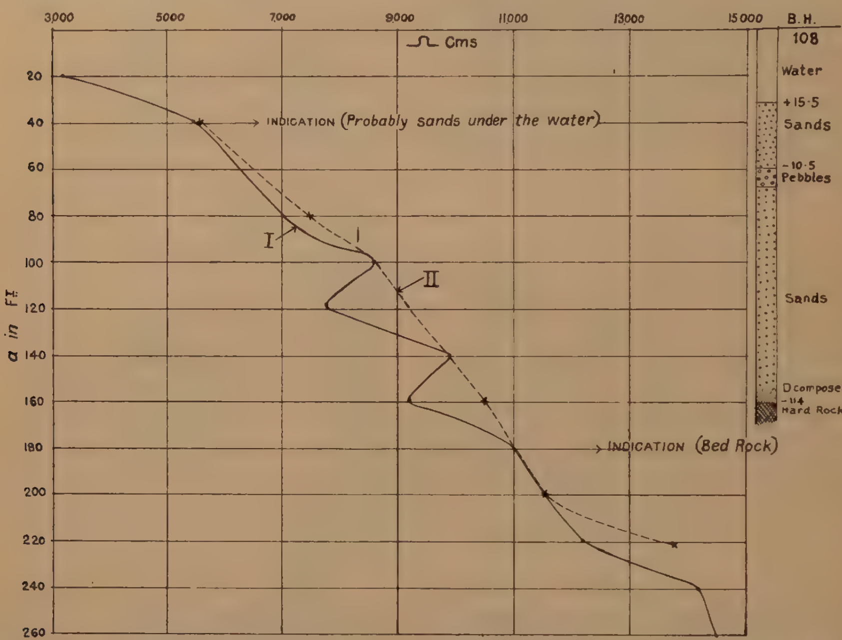


FIG 18—EXPANDING ELECTRODE TEST NO. 20. WATER COURSE, MAHANANDISWARAM EAST AREA (Y-Z_1).

by four swimmers who were each equipped with a life belt. These swimmers, directed from the two mobile boats, moved on to occupy the desired positions for obtaining the required electrode separations. The contacts in water were made by directly dipping the metallic strands of the V.I.R. cable leads, with the ends freed from the insulation for about 9 in. The contact resistances with such an electrode system were about 500 ohm (for both the potential electrodes) and usually did not vary much from this value at the several positions because of the uniformity of the contacts in water.

First of all, a series of readings were taken in different parts of the water course by adopting a fixed electrode separation of only 5 ft. This was done with a view to noted to have a constant value of about 3400 ohm-cm.

The expanding electrode test was then carried out, taking measurements from 20 to 280 ft (the largest possible) increasing by successive steps of 20 ft. The test could be only across the river, in an east-west expanding system, as it was found difficult to maintain an alignment correctly parallel to the water course. In order to obtain a check on the first set of measurements, a second set of readings was taken by "converging" the electrodes back in steps of 40 ft. The two sets of resistivity values so obtained are graphed in the usual way and shown in Fig 18. There is good agreement between these two curves until 100-ft separation. In curve 1 (Fig 18) taken in steps of 20 ft, there are two discrepancies,

one at 120 ft and the other at 160-ft separations. If we smooth off these discrepant values on the first curve, the agreement between the two will be obvious.

FIG 19—MEGGER EARTH TESTER SET UP ON TRIPOD.
MahaNandiswaram Temple Island (Y) in the background.

Empirically, two indications are recognizable in these curves: the first is at 40 ft and is perhaps to be attributed to the upper interface of the sandy bed of the water course; the second is at the 180-ft separation. The resistivity gradient falls a little after 160 ft, and shows a much greater increase after 200 ft. The key indication is not clearly noticeable in this case, but the anomaly, either at 160 or 200 ft may be used as the indication. Taking the geological probabilities into consideration, a value between these, 180 ft, was put down as the depth to bedrock. Treating the curve as a simple two-layer case, the logarithmic fittings denoted 176 ft as the depth to the resistive bed. From these considerations the depth to bedrock in the middle of the water course was estimated to be 180 ft or about −140 ft m.s.l. and, the geophysical indication clearly pointed to the existence of a deep section beneath the area covered by the water course also. The writer has recently been informed by the chief engineer that two bore holes (B.H. No. 108 and 113) were drilled in the water course between Y-Z_1 (Fig 6) and rock was met at depths of 160 and 147 ft, respectively. So the geophysically estimated depth of 180 ft is in error by 12.5 pct. It would now

FIG 20—MEGGER EARTH TESTER IN OPERATION.
Village of Ramayyapeta and the hill (X) on the right bank, in the background.

appear that the anomaly at 160-ft separation more closely corresponds with the actual depth noted in bore hole No. 108. Decomposed rock for a thickness of 4 to 5 ft above the hard bedrock has also been recorded in the bore hole results, which explains the key indication.

VERIFICATION OF THE ELECTRICAL INDICATIONS BY DRILLING

Although the operating principles of the interpretation of the resistivity curves appear at first sight to be simple and easy of application, actual experience shows numerous complicating conditions. In many cases, it was found exceedingly difficult to offer any useful estimates of the depth to bedrock without having recourse to the empirical method in some manner or other. Perhaps it might appear irrational that in estimating the depth reliance has been placed on one method of interpreting the curves in one case and another or a third method in another case. Under the complicated set of conditions prevailing in the area, a rigid, mechanically applicable

method was not only impracticable but also would have led to serious mistakes in the estimation. By a very careful and comprehensive appreciation of the geological probabilities a reasonably correct interpretation could be built up. A few cases, however, proved wholly unsuitable. These were in the high-level terrace areas where highly conductive, thick clay beds constituted the surface formation and the insulating sands and pebbles underlying them behaved as possessing infinite thickness and masked up the indications of the bedrock.

However, in a fairly large number of cases in other portions of the river bed, where, despite the variations in the composition of the alluvial materials and the irregularity in their stratified structure, a favorable condition of the bedrock has helped in the interpretation. In such cases, the bedrock was decomposed for about 2 or 3 ft below the overburden. This thin decomposed material constituted a highly conductive layer, causing a peculiar anomaly on the curves, and making it possible to estimate, empirically, the depth to the bedrock. Judging purely from the results obtained in the area, this method of following up the key indication, and also Irwin Roman's method of logarithmic-curve fittings in some appropriate cases, have proved definitely serviceable in affording an insight into the subsurface-rock topography and aiding in the location of bore hole spots calculated to obtain the most informative data for expeditiously arriving at a decision on the relative *prima facie* merits of the sites.

Of the 26 expanding electrode tests carried out a verification by borings, at or very close to the test spots, is now available for 14 of them. Of these 14 cases, the estimated depths were "announced" before

TABLE 1—*Tabular View of the Electrical Indications*

Serial Number of Field Curves	Nearest Bore Hole Number	Location	Electrically Estimated Depth to Bedrock, Ft	Actual Depth in Bore Holes, Ft	Percentage of Difference	Type of Curves
Ex. 1	7	High level terrace, *IA* Site	90[a]	202.57		Type 1
2	23	Sandy bed, *α* Site	160	175	−8.6	Type 4
3		High level terrace, *IA* Site	120[a]			Type 1
4		High level terrace, *IA* Site	140			Type 2
5	2	Sandy bed, *IA* Site	100	99.4	+0.6	Type 1
6		High level terrace, *α* Site	120			Type 2
7	27	Sandy bed, *IA* Site	120	132.5	−9.5	Type 3
8	29	Sandy bed, Ramayyapeta Site	100	92.5	+8.1	Type 2
9	30	Sandy bed, Ramayyapeta Site	100	98.45	+1.6	Type 2
10		Sandy bed, Ramayyapeta Site	160			Type 3
11		Sandy bed, Ramayyapeta Site	180			Type 3
12		Sandy bed, Ramayyapeta Site	160			Type 2
13	36	Sandy bed, Ramayyapeta Site	200	205.6	−2.5	Type 2
14	37	Sandy bed, Ramayyapeta Site	120	111	+8.1	Type 2
15	34	Sandy bed, Ramayyapeta Site	140	153.1	−9.4	Type 2
16	33	Sandy bed, Ramayyapeta Site	80	43.1[b]	+85	Type 2
17		Sandy bed, Ramayyapeta Site	140			Type 2
18		High level terrace, Ramayyapeta Site	60[a]			Type 1
19		High level terrace, Ramayyapeta Site	60			Type 1
20	108	Water course in Y-Z_1 portion	180	160	+12.5	Type 1
21	28	Sandy bed, V Site	180	121.5	+48.6	Type 4
22		Sandy bed, V Site	200			Type 5
23		Sandy bed, V Site	160			Type 5
24		High level terrace, V Site	120[a]	220		Type 1
25	21	Sandy bed, VI Site	140	Rock not touched at 143	?	Type 2
26		High terrace, Sites V and VI	80			Type 1
SP-I	6	Sandy bed, *IA* Site	190	186	+2.2	Type 5

[a] Indications refer to depth of insulating sandy beds beneath clays rather than to bedrock.
[b] Bore hole has touched locally high peak of subsurface rock. Other holes drilled in area show rock at about 80-ft depth.

the drilling in 11 of them and the remaining three were tried close to the bore holes that had already been drilled. Regarding the balance of 12 unverified cases, in at least six of them it is known that the geophysical estimate is quite in accordance with the subsurface conditions either already established by borings in similar parts of the river on other adjoining alignments, or are reasonably to be expected from the circumstantial evidences available.

A summary of the electrical indications obtained in these investigations and the results of the verification, for the available cases, are given in Table 1. For the verified cases, the difference between the electrically estimated depth and the actual depth is expressed as a percentage of the latter. It may be seen that the differences so noted are within about 10 pct in the majority of the verified cases.

Acknowledgments

The writer desires to acknowledge his indebtedness to all those who have helped him in this investigation. Dr. M. S. Krishnan, as the Superintending Geologist in charge of the Madras Circle and as the author of the proposal for the application of the geophysical method in this area, has contributed a great deal to this investigation. He not only spent some time in the field with the writer, giving many valuable suggestions for the conduct of the work, but also kept an unfailing interest throughout the survey, guiding and helping in several ways. It is hardly possible to make an adequate acknowledgment of the writer's indebtedness to him. Mr. R. N. P. Arogyaswamy assisted in the field and in interpreting the curves, and finally in the preparation of the report. It is a pleasure to acknowledge his assistance.

The writer's thanks are also due to Diwan Bahadur N. Govindaraja Ayyangar, Chief Engineer of Madras, and also to Messrs. M. K. N. Pillai and R. Sivasubramanyam, Engineers in charge of the Project, for furnishing the bore hole data from time to time. For permission to contribute this paper, making use of the material from the report which the writer had submitted in 1944, thanks are due to the Special Chief Engineer to the Government of Madras.

The illustrations for this paper have been drawn by Mr. K. Sankar Rao, formerly of the Mysore Geological Department, and the writer would be failing in his duty if this kind service were not acknowledged.

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DISCUSSION

F. W. Lee:* A careful review of this paper on the selection for a dam site shows that the geological features which control the situation have been clearly understood before the geophysical work was planned, as it should be. With this information, the geophysical planning for measuring the topography of the bedrock clearly showed that shallow seismic refraction methods would be most suitable for this geological condition since only slow speed beds covered the crystalline high speed bedrock.

However, owing to the great expense of seismic field equipment, resistivity methods would be the second choice for the complicated geological condition in this area. Mr. Rao has made a very careful electrical survey and has analyzed the different electrical subsurface con-

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ditions in a very detailed manner, and the information which he obtained, both in a positive and a negative sense, greatly enhanced the base of information for planning this project, which good geophysical work always does. Mr. Rao has shown a very high appreciation of the many factors which must be taken into consideration in interpreting electrical resistivity surveys, which are generally not considered of much importance from a purely geological point of view but seriously modify the interpretation of the electrical subsurface characteristics. In particular, I refer to his mentioning the key horizon composed of a geologically insignificant bed of decomposed rock, which had become electrically, relatively highly, conducting, and the use of this horizon in making proper interpretations.

This contribution is particularly valuable since it clearly shows what information can be obtained on dam sites when only electrical methods are applicable as would be the case in many flood control projects where only sedimentary formations abound.

R. W. Moore*—It is of considerable interest to learn of the very practical use which the Government of Madras, South India, is making of earth-resistivity surveys. Although much has been done in the United States in applying this relatively rapid and inexpensive method of test to a variety of engineering problems during the past fifteen years, insufficient publicity has been given the results obtained. We are indebted to the author for making public the results of his tests at an early date following their completion.

Mr. Rao's use of so-called "key indications" to locate possible changes in the substrata appears to have merit. Certainly, where such characteristic inflections occur in the curve in close agreement with subsurface changes found with the drill, their use as demonstrated by the author would seem to be a valid procedure. His willingness to employ several different methods for analyzing resistivity curves, using one when the others are not applicable, or using two or more methods as a check is worthy of note.

Mention is made of certain data obtained that could not be properly analyzed by any of the methods employed. In the relatively shallow tests characteristic of highway exploration problems, such as have been under study by the Public Roads Administration, it has been demonstrated that refraction seismic tests often can be used to supplement or augment data obtained with the resistivity apparatus. Use of the seismograph to obtain data in verification of results obtained with the resistivity test is possible because of the entirely different properties of the earth's materials that are involved in the two methods of test. In cases where one type of test gives indications that approximate closely those obtained with the other method the need for the drilling or boring of test holes may be reduced or even at times eliminated. In the work reported by Mr. Rao refraction seismic tests probably would have been of considerable value as a supplement to the resistivity tests.

Use of the resistivity traverse or constant-depth test in a study of the lateral changes in the substrata is aptly demonstrated by the author. Similar applications are to be found in the relatively shallow tests that have occupied the writer's attention for some years. Having established the existence of an anomaly such as a buried river channel or some other subsurface structure, its location may be traced over wide areas. In highway work such tests are useful in locating sand, gravel, and solid rock for construction purposes.

The author mentions the cumulative resistivity method of plotting proposed by the writer and states that further work is contemplated in studying its possible usefulness in interpreting data he has on file. Publication of any correlations that he may obtain between this and other methods would be a valuable contribution and may help in making future resistivity surveys appear less formidable particularly with regard to the interpretation of the data obtained. It is to be hoped that this paper will encourage others to make public the results of other field studies and the methods used for their interpretation.

Mr. Rao's description of the limited amount of testing done over water-covered areas is of particular interest in that it serves as a reminder that such tests are not only possible but practical.

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H. C SPICER*—The paper by Mr. Rao is well arranged and presented, both in text and figures. The introductory remarks and the sections giving pertinent descriptions of the topography, geology, and sites investigated give an excellent background for the geophysical problem. They also indicate that the author has a clear understanding of the area under investigation. The sections following in which methods, apparatus, and interpretations are discussed do not, I regret, similarly impress me.

During Mr. Rao's visit to this country last year, he arranged to spend about a week with me both in the field and in the office. Most of the points given formally here were discussed in person with him before I had read or heard his paper.

To make resistivity measurements entirely worthwhile, it is, in my opinion, necessary to adopt an entirely objective method in the interpretation of resistivity curves. This statement implies that the resistivity curves will be evaluated by accepted physical and mathematical methods, and that the resulting interpretations will be correlated with the drilling logs and other geological information available. A uniform application of this technique will minimize the personal factor. Interpretations of resistivity curves based on "experience and geological probabilities," as Mr. Rao states, seem to put them too near the "forked twig" category.

In the examples of apparent resistivity curves given in this paper, the shallow observations from 0 to 20 ft to a maximum of 0 to 60 ft apparently are not considered in the interpretations. Such technique will not yield a correct interpretation of the resistivity curve, as the resistivities of the upper layers and their thicknesses must be considered. These factors are fundamental in the evaluation of resistivity curves.

Mr. Rao's attention is again directed to the papers by Hummel,[10] Watson,[11] Watson and Johnson,[12] Wetzel and McMurry,[13] and other papers not specifically mentioned here; papers giving an objective treatment to the interpretation of resistivity curves.

I fully recognize the inability of one not present during the taking of the observations to criticize justly and evaluate the resistivity curves. Nevertheless, it seems to me that at least some of the ragged curves obtained by Mr. Rao may have resulted from faulty operating technique. Seemingly his types 1, 2, and 3 should be three-layer or perhaps four-layer curves. In his type 1 where there is a low resistivity surface layer, his curve is smooth but in his type 2, where the surface layer is high, the curve is ragged beyond the 80 ft depth. My experience has been that when a thick, very low resistivity layer lies below an extremely high resistivity surface layer, it is almost impossible to obtain a smooth curve below the second layer unless very heavy currents are passed into the earth. A high surface layer is not naturally adapted to passing such large currents easily, and as a result of using low currents the curve takes on, as Mr. Rao puts it, a "rather simmering fluctuation" which is confusing and meaningless to one using objective techniques of interpretation. Likewise, his types 4 and 5 seem to indicate a similar fault.

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M. B. RAMACHANDRA RAO (author's reply)—The author is thankful for the kind reception accorded to this paper both at the meeting where it was presented and in the written discussions contributed after its publication.

As pointed out by Dr. Lee and Mr. Moore, the seismic method surely would have been the most suitable one for the investigation. As there was no seismic equipment, the electrical resistivity method had to be used. The author has since had several opportunities during his stay in the United States and Canada last year for studying the excellent geophysical investigations conducted by the government departments and by some private organizations. He has also noted the types of new portable equipment which have come into use for shallow refraction shooting problems. In particular, the author has studied the Shepard type instrument which is so useful in solving bedrock problems in highway and other engineering projects. Attempts are being made to introduce such seismic methods in India.

Mr. Spicer's comments are valuable reminders of the pitfalls in the use of the electrical resistivity method. Many a source of error which may creep into measurements had also been kindly pointed out to the author during a visit with Mr. Spicer's field party and

the author derived considerable benefit from that visit and personal exchange of views with him. However, to enter into a detailed discussion of the substance of his present remarks on the interpretation of the curves would put one in the awkward position of having to dig out for an airing all the skeletons of the controversies on the empirical vs the theoretical methods. The author fully shares with Mr. Spicer the view that an objective method of interpretation should be adopted. But, it is no objective method to ignore indications which are found reliable and valid in actual practice. Faults in the operating technique might have caused some of the ragged features in the curves, but the indications relied upon by the author for the interpretation are actually independent of such factors. Measurements were made by expanding the electrodes along two mutually perpendicular directions in each case and often the Lee-partitioning was also adopted: the several meaningless fluctuations in the curves due to poor ground contacts, surface irregularities, and other similar effects have not been used, nor any of them selected aribitrarily as a key indication. Only the anomaly which was corroborated in the two or more sets of curves of the particular case, was considered. Moreover, this indication was a feature which qualitatively could be expected and in a sense, even reasonably explained on the basis of the prevailing geological conditions.

Many instances could be cited where a "key indication" similar to that used by the author has been successfully employed by others under quite different conditions. Dr. Swartz's curves[14] contain features which are similar to the key indication used by the author. There are also a large number of curves obtained in shallow overburden in different parts of the world, in which sharp anomalies have been found approximately corresponding to a subsurface geological discontinuity. Surely, every one of those cases cannot be said to be the result of faulty observations.

Mr. Spicer in expressing his distaste for basing interpretation of the curves on "experience and geological probabilities" mentions the "forked-twig" category. However, the meaning which the author has associated with the word "experience" is nothing but the practical acquaintance gained by long and varied observations. Geological probabilities cannot be left out of consideration in any of the geophysical methods, if the interpretation is to be of any practical value.

The subjective factors like assumptions, selection, and judgment are not dispensed with even in the theoretical methods, including those Mr. Spicer himself uses and recommends. In the concluding sentence in their paper, Wetzel and McMurry[13] say: "Finally, it is not claimed that this interpretation method may be used automatically to grind out answers. Judgment in the application of resistivity methods are still required in order to obtain even the optimum 10 pct accuracy." The mere fact that an observed field curve matches perfectly with one or the other of the curves worked out on theoretical assumptions, is in itself no positive proof that the actual conditions would correspond to the assumed conditions. The solution based on a simplified physical and geometrical concept could be used only with proper appreciation of the geological probabilities, and experience would in any case be necessary to offer a reasonably correct interpretation. There is nothing unscientific in such methods or procedures. Just because there is an element of subjective consideration, this complexity should not be misunderstood.

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Health and Safety in Operations of the Consolidated Coppermines Corporation

By B. P. Burt* and E. B. Olds,† Member A.I.M.E.

(Chicago Meeting, February 1946)

The mines of the Consolidated Coppermines Corporation are at Kimberly, in the Robinson mining district, White Pine County, Nevada.

Mine openings in either type of porphyry require heavy timbering and constant maintenance and repair, particularly in horizons at or near the ore body contact.

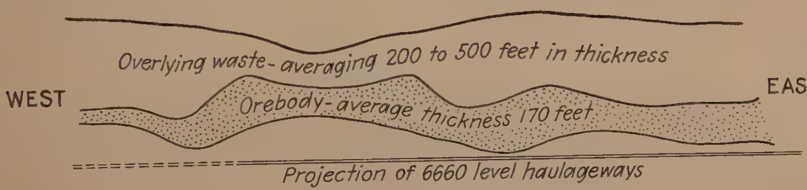

Fig. 1.—General section of Emma Nevada ore body through long axis.

The blanket-like ore body is a disseminated copper deposit occurring in monzonite porphyry and, in general, is thoroughly fractured, altered, and soft. Contained in the porphyry ores of the extreme westerly end of the ore body are pendants of hard garnetized limestone. A younger monzonite porphyry underlies the ore-bearing porphyry, which in turn rests on limestone and shale.

Block caving is the standard method of mining. Because of the undulating top and bottom contours of the ore body, the main haulage level was selected and driven at an elevation calculated to be the most economic depth below the major tonnage sections of ore. Fig. 1 shows a general section of the Emma Nevada ore body taken through its long axis, which lies roughly in an east-west direction.

Where the ore lies close to the haulage level, slusher hoists and scrapers are used to transport horizontally the ore to the main haulage drift (Fig. 2). For the extraction of high-lying ores, a system of branch raises is used; and the ore is mined by gravity. Slushing methods are used also on the high-lying ore-body fringes. Slusher drifts are often driven from the fringe branch raises that have already served their purpose, in order to minimize the cost of the preparatory work in mining the thinner portions of the ore body (Figs. 3 and 4). When a block of ore is developed and undercut, it is economical to mine it as rapidly as the ore will cave efficiently; and this procedure often presents safety problems because of men with equipment working in a relatively congested area.

The single-track main haulage drift is about 1½ miles long, and lateral haulage drifts are driven under the ore body at intervals determined by the height of the ore body above the haulage level. In order to minimize repair and maintenance, the

Manuscript received at the office of the Institute Jan. 31, 1946. Issued as TP 2049 in Mining Technology, September 1946.

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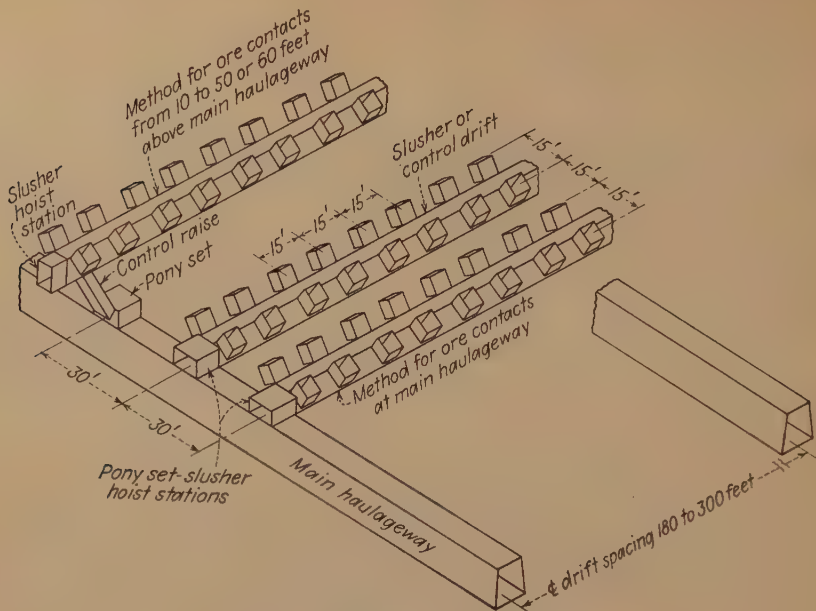

Fig. 2.—Main haulage drift.

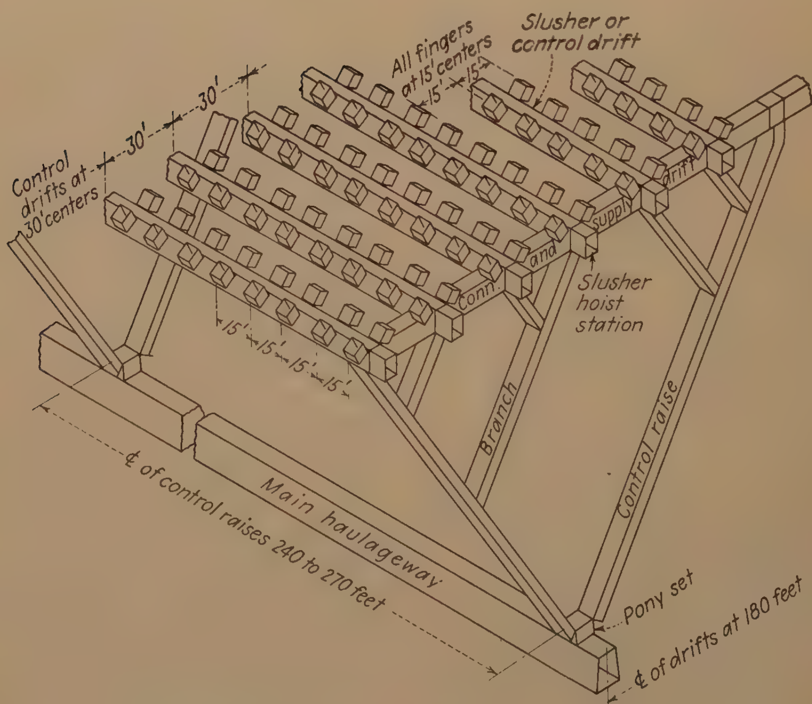

Fig. 3.—Standard slushing method for high-lying ores.

haulage drifts are driven and timbered as small in cross section as is possible to satisfactorily operate the haulage equipment. In most timbered sections the haulage drifts are timbered with 12-in. Douglas fir. The caps are 8 ft. 6 in. long, with posts 8 ft. 8 in. and 9 ft., framed to have a 2-in. batter per foot. The 9-ft. post is on the drainage-ditch side of the drift.

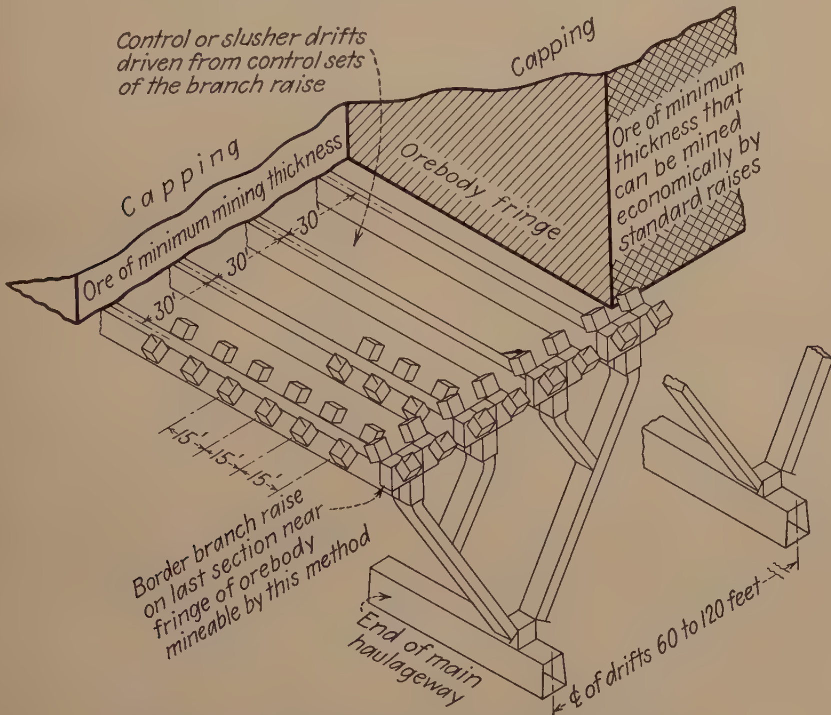

FIG. 4.—SLUSHER MINING AT FRINGES OF ORE BODY.

The main hoisting shaft, Emma Nevada, is divided into five compartments, having an over-all cross section of 32 ft. by 7 ft. 6 in. The cage is double deck, each deck holding 25 men. Two skip compartments are used; each skip weighs 12½ tons with a capacity of 12½ tons of ore. The manway compartment is utilized for the main electrical conduits, air and water lines. A small compartment is used for the counterbalance of the man hoist.

The tracks in the main haulage drift are 75-lb. rails and in the lateral haulage drifts, 60-lb. rails. Electric trolley locomotives are used, of 10 and 13 tons weight. The ore is transported in 5-ton capacity bottom-dump cars. The bottom-dump cars were selected because the design permits the handling of maximum tonnages in the smallest cross section of haulageway.

The movements of all trains are under the direction of a centrally located dispatcher. When a train is loaded and ready to go to the station for dumping, the motorman signals the dispatcher by a push-button switch in the lateral drift near the approach to the main haulage drift. This switch lights up a frosted glass in the dispatcher's office, indicating the number of the lateral drift in which the train is loaded. When the track is clear for the particular train to go to the station, the dispatcher signals, by code, by the use of a

push-button switch, which gives the motorman a series of flashes on light globes near his own push-button switch. As the train returns from the station and is in its lateral drift, clear of the main drift, the motorman indicates this by giving the dispatcher a code signal on his push-button. All signals between motorman and dispatcher must be repeated by the receiver. The dispatcher, through the miniature block-lights on the panel before him, is able to follow the position of all trains along the main haulage drift. As a train goes into a block, it automatically turns on the block-lights in the drift by a trolley contact switch, which energizes a set of alternating-current relay controls that turns on the proper lights and the corresponding lights on the dispatcher's panel or map of the mine. A loaded train going to the station (easterly) turns on blue block-lights in the drift and on the dispatcher's map, while a train returning to its lateral drift (traveling westerly), turns on amber lights. The colored directional block-lights provide full protection because if the dispatcher should release a train by mistake, the motorman can detect an oncoming train and avoid collision. The dispatcher, by turning on a block-light, is able to stop or release trains going in either direction at various points in the main haulage drift.

A train crew consists of a motorman and a brakeman; the latter rides in the last car on the train. These men are able to signal each other by the use of light-boxes and push-button switches. Each car is equipped with a three-conductor rubber-covered cable on which are attached convenient connectors at each end, enabling the trainmen to make a quick connection when cars are coupled, thus providing for two-way signaling on the train.

Organization

The Safety Director and Inspectors are responsible for the purchase and maintenance of most of the safety equipment and supplies, inspection as to health and safety of the plant (both surface and underground), equipment, machinery, good housekeeping, safety guards or appliances, warning signs, parking, traffic, and other matters concerning the general physical welfare of all employees.

Several safety meetings are held during each month. The schedule for these meetings is prepared on a quarterly basis. A list of the day-pay employees who are to attend the employees' meetings is currently prepared and the attendance is rotated in order that an employee may have the opportunity to attend at least one meeting during a calendar year. The department heads, foremen, and all other bosses are required to attend a bosses' meeting held at least once during each calendar month. The types of meetings are divided into three general groups: (1) meetings attended by employees only, and conducted by the Safety Director and his assistants; (2) bosses' meetings, attended only by the bosses; and (3) meetings attended by the department heads and their assistants and other bosses that the department head may designate. During the quarter at least one meeting is attended by all department heads, foremen and bosses, and by the General Superintendent and General Manager. The General Superintendent and General Manager occasionally attend one of the other meetings, whenever some special reason for attendance occurs. It has been found that much more can be accomplished in these meetings when the attendance is not too great. It has also been found that special meetings are desirable for the employees of only one department, in order to take up some particular matter peculiar to that department. For instance, the underground trainmen meet once a month to discuss safety matters pertaining specifically to their work, Morris employees meet once a month and Emma employees once a month.

At these meetings all subjects pertaining to health and safety on the job are discussed. These include talks on the symptoms, transmission and treatment of social and industrial diseases, off the job safety, safety in the home, and other important subjects. Suggestions on elimination of hazards, working conditions, health and safety may be brought up and discussed. Any pertinent subject may be requested for study and discussion at the next regular meeting.

Employees' meetings are held on company time at the beginning of the shift. Although no time limit is set, the meetings are adjourned when the first sign of lack of interest appears or the discussion lags.

The Safety Director studies each subject to be discussed on disease and health by consulting medical journals, papers and books, under the supervision and with the help of the Corporation's medical staff. The Chief Surgeon usually prepares a syllabus or short paper on some subject to be presented for talk and discussion. All nontechnical information in hand is kept in the Safety Director's files and made available to those that are interested.

The bosses' and department heads' meetings include refresher courses in the use of the resuscitator and inhalator, oxygen breathing apparatus and safety equipment. Accidents and safety suggestions are reviewed. Each department head is asked what disposition was made of the suggestions pertaining to his department and what corrective actions have been taken since the last accident that occurred in his department.

Suggestions on health and safety are obtained during the regular meetings, through suggestion boxes and by personal contact on or off the job. After the suggestion is obtained, it is the duty of the Safety Department to contact the department head involved, discuss the problem, arrive at some possible solution, and inaugurate measures for correction immediately. Occasionally a suggestion presented in good faith by an employee may be entirely impractical and proof on that fact may be offered, during an employees' meeting, to the satisfaction of the person who made the suggestion.

Suggestions that may still be in the category of impracticability may get beyond an employees' meeting and be presented to a particular department head, who may satisfactorily prove to the Safety Director and the person making the suggestion the futility, or impracticability, of the suggestion, or he may take up the matter up in a bosses' meeting. If no satisfactory solution or conclusion is reached, the matter can be taken up with the General Superintendent. If the suggestion is not followed, the person making it is always sought out, generally by the Safety Director or the particular department head involved, or by both, and a satisfactory explanation is given for the reason the suggestion could not be followed. The Safety Department notes the completion of the suggestion, observes the results and reports at the next series of meetings. A monthly summary of all meetings and suggestions is published, listing the names and pay-roll numbers of the men present and brief statements of suggestions offered. This is sent to each department head and boss and displayed on all bulletin boards. A copy is available for any employee interested. Absentees of the supervisory staff are noted for permanent record.

Courses in First Aid, and Mine Rescue, under direction of the United States Bureau of Mines, are made available to every employee. Special First Aid courses are given for the bosses, and upon completion of these courses these men are qualified for the Provisional Instructor's Certificate from the United States Bureau of Mines and the Red Cross.

Safety Equipment and Supplies

The safety equipment and supplies placed at strategic locations throughout the

FIG. 5.—UNDERGROUND FIRST AID STATION AT EMMA NEVADA MINE.

FIG. 6.—UNDERGROUND FIRST AID STATION AT MORRIS BROOKS MINE.

underground and surface plants include: resuscitators and inhalators, gas masks, trolley protectors, self-contained oxygen breathing apparatus, life lines, stretchers, first aid supplies (Figs. 5 and 6) safety belts and fire equipment. Several racks (Fig. 7) of safety pipe spiling, special drilling-machine bits for driving the spiling, bars and other tools, all painted red, at convenient places in the mine, are for use in emergencies only; for instance, if men are trapped underground.

Underground employees must supply themselves with hard hats, safety shoes and waterproof matchboxes. Most underground employees use electric cap lamps and the Corporation furnishes small candles, which all employees are required to carry underground for use as a sensitive indicator of air low in oxygen content. Equipment given to employees without charge includes: many types of safety goggles and respirators, individual first aid packets for use during working hours, and first aid supplies for the home.

First aid supply cabinets (Fig. 8) are distributed and maintained at handy and convenient points throughout the entire plant. The practice at first was to lock the cabinets and give the key to the

boss in charge of the particular section of the mine or plant. When an accident occurred, it was often necessary to break the locks or boxes to meet the emergency and soon afterward most of the supplies would vanish. Later it was decided to remove the locks and make the boxes accessible to everyone alike—but still the supplies disappeared. Finally, it was announced at all of the safety meetings that first aid supplies would be furnished to all employees for use in the home if they would report to the Health and Safety Department. This idea produced gratifying results, for (1) within a few weeks the pilfering of supply cabinets practically ended, (2) total cost of such supplies dropped almost 60 per cent.

The Consolidated Coppermines Corporation maintains a modern 26-bed hospital (Fig. 9) fully staffed with physicians and surgeons, nurses, and other necessary hospital personnel. A doctor is available 24 hr. a day, and may be called to any working place on the surface or underground when the occasion demands his immediate presence. The Company ambulance is also available at any time during the twenty-four hours.

Fig. 7.—Racks of tools painted red, for use in emergencies.

Safety Devices and Practices

A two-way "teletalk" has been installed between the ore hoistman and the skip loader which enables them to freely talk to each other while they are actively engaged in their work, but does not replace the electric bell and flash system for the control of the movement of skips. This also enables the hoistman to know immediately if anything is wrong at the loading station

Fig. 8.—Trolley locomotive in lateral drift. First-aid supply cabinet on post.

when he hears no word or noise of any sort over the "teletalk" system.

Several safety devices are provided for the large ore hoists and man hoists for protection against overwind, underwind, overspeed and protection if anything should happen to the operator while the hoist is in motion. In the year 1928, the electrical and mechanical departments carried on experiments with photoelectric relays for the purpose of providing means of cutting off power and stopping hoists that were in motion within the narrow working limits in the headframe. It was found that the development of cells used in the relays at that time had not reached a point where they could be depended upon. However, in subsequent years, after further development of the photoelectric cell, a new series of tests was carried on, and it was found that the relays could be used as efficient safety devices. All of the large hoists are equipped with "Lilly" controls for protection against overspeeds, overwind and underwind and photoelectric cells have been installed for many years as an additional safety factor just beyond the working limits of the Lilly controls. These installations were made to prevent serious wrecks in the event that the Lilly controls did not function because of mechanical failure.

Another device installed on the large hoists is referred to as an "interlocking relay," which operates only if the skip or cage has gone through the Lilly control and through the photoelectric cell limits, and the hoist is stopped and locked. Usually, when such an incident happens, the operator becomes slightly confused, and after he has thrown the back-out switch there is a chance that he will apply the power in the wrong direction. The interlocking relay device is so connected to the operator's control that he can apply power only in the right direction.

Many electric slusher hoists are used in the underground operations. The mine atmosphere is damp, and early experiences showed that the ordinary safety switches were not good enough to prevent possible short circuits and blow-ups of the switch, with consequent serious accidents to the operators and a possible source of mine fires. The conventional type of safety switch often short-circuited because of the absorption of moisture into the material that composed the block on which the switch was mounted. A combination switch was then developed, which provides a disconnect switch, fuse protection, thermo-protection and magnetic contactor. This permits the placing of the combination switchbox at a safe distance away from the operator and allows him to turn the current on or off with a push-button control within arm's reach of his position while operating the hoist. The character of the materials used in the switch and the method of housing have proved entirely satisfactory and have led to a minimum of delays without serious accidents.

Luminous paint is used for outlining the walkways in the hoist houses and other plant installations, particularly near high-voltage electrically operated equipment. This innovation was introduced during the war period, when blackouts were deemed advisable. If the light fails, the paint provides a clear outline of the walkways, and prevents accidents caused by men walking into dangerous objects.

The safety dogs on all man and material cages are tested regularly once a week. To facilitate these tests the mechanical department has devised short guides, placed just above the collars of the shafts, which may be removed quickly after the tests have been made. A careful inspection is made of all major hoisting cables at least once a month. The diameter of the cable is measured in a number of sections, in order to determine the amount of wear, and measurements are also taken for the length of wear on the outside strands. By these inspections and measurements it is possible

to calculate, with reasonable accuracy, the remaining percentage of strength left in the cable.

The cages in the main hoisting shafts are at times pulled up through the safety limits of the Lilly and photoelectric cell, in order to permit 30-ft. long rails to be attached and lowered underneath the cages. When this operating procedure becomes necessary, an electric safety device rings a bell and lights a warning sign, which reads "Danger—Cage Too High," in front of the hoist operator. The lights remain on and the bell continues ringing until the cage is lowered through the safety limits and reaches the safe operating position in the shaft.

Underground trolley locomotives painted yellow, with a "Safety First" sign painted on each side, loom up better than those painted in any other color.

Whether an employee is working in an underground or a surface shop, he is not permitted to grind, or work in any manner where there is a danger of flying metallic chips, without wearing safety goggles, which are furnished by the Corporation.

Rubber trolley-wire covers are required wherever any repair work is done around the trolley wire on the haulage level.

A part of the top edge of the trailing end wall of the last ore car on each train is cut off, on which a 2-in. pipe is welded. This arrangement enables the brakeman, who rides in the last car, to get in and out with greater ease and safety when it is not loaded.

As stated, the ground, in general, is "heavy," and openings require heavy timbering and much repair for proper maintenance. For cover lagging and side lacing the best quality of seasoned red fir "split lagging" is used. This timber is obtained from the Pacific Coast areas. The lower parts of the fir trees are cut in the desired lengths and split concentrically, so that the fibers usually follow the full length of the stick. The lagging timber prepared in this manner bends considerably and even partly breaks before complete failure (Fig. 10). This flexibility gives the miner ample warning before there is a serious cave-in. The average dimensions for this type of split lagging are 2½ by 6 in., in lengths of about 5 feet.

The ground at the mining or drawing horizon is particularly heavy, and sometimes for additional strength and stability two or three pieces of split steel rail are alternated with the split lagging for the protection of the cover of the mine opening. The split rails are obtained from the salvage 60-lb. and 75-lb. rails used on the haulageway. The rails are cut in the desired lengths and then cut into two pieces, with a torch along the web. The ball half and the flange half of the rails are equally good for this usage.

FIG. 9.—PART OF HOSPITAL PLANT. HOSPITAL IN BACKGROUND, NURSES QUARTERS IN FOREGROUND.

At the top of all transfer raises safety doors are installed. One door covers the ladder of the manway, and another door covers the timber slide. When closed, the two doors meet in the middle of the manway. The doors are constructed with a 1-in. pipe frame. The frame is covered with a heavy-gauge 1-in. mesh screen. This makes a strong, light door, which does not hinder ventilation.

FIG. 10.—SPLIT LAGGING IN DIFFERENT PHASES OF BENDING AND BREAKING.

Timber and supplies are hoisted up raises with light air tuggers. The raises are equipped with two-way flash-signal systems.

VENTILATION AND FIRE CONTROL

Induced ventilation, for the main part, is accomplished by two reversible fans at the top of ventilation raises. One has a capacity of 50,000 and the other 44,000 cu. ft. of free air per minute. In dead-end drifts or raises, ventilation is augmented whenever necessary by the means of small fans of 1200 cu. ft. capacity of free air per min. through 12-inch diameter flexible or metal conduits. These fans are small enough to be installed in any haulage drift or convenient opening, and still be out of the way. In dead end slusher drifts an air induction jet is usually used.

With fire doors strategically located along the main haulage drift and in the ventilation drifts used in conjunction with the reversible fans, it is possible to have the flow of air directed either upcast or downcast in any of the six shafts or raises connected to the surface. Carbon dioxide or the foam-type fire extinguishers are kept in all underground shops and around electrical machinery. Water pressure at 140 lb. per sq. in. is maintained on the main haulage level. There are water connections at convenient places throughout the entire mine workings. A supply of water hoses is kept in all underground tool rooms.

For surface and town-site fire protection, carbon dioxide, foam or pyrene-type extinguishers are available in all major plant buildings. A well-equipped fire truck is maintained for immediate use, and three trained volunteer fire-fighting crews.

RECORDS

Experience has shown that all accident reports, statistical data and records pertaining to safety should be kept in as simple a manner as possible in order that they may be easily understood for current

or future reference. Such reports, data and records should show concisely the source of trouble or accident, and point the way for immediate correction and/or elimination.

These reports include: (1) the foreman's detailed report of the accident, (2) the monthly summary of all accidents by departments, (3) each individual's file and case, (4) his length of service before injury, (5) the part of the body affected, and (6) monthly and yearly reports to the management.

In reporting length of service before injury, the employees are divided into groups by number of years of service and the rates are calculated against percentages of these groups to the total payroll. The lists of the accident-prone group or groups show up first for reference and study and then for further instruction.

1. The Foreman's Report of Injury to Employee gives; name of injured person, his occupation, check number, rate of pay for 8-hr. shift, date and time of accident, place accident occurred, nature of injury, name of shift boss on duty, names of witnesses, names of first arrivals, description of the way in which accident occurred, and date of reporting accident.

2. The Accident Summary (monthly) includes: name of boss, name and number of employee, name of department, working place, description of accident, extent of injury, and date.

3. Injury Record includes all items of Accident Summary except boss's name; and, in addition, the date workman was hired, days charged, and type of accident (fatal, permanent, temporary, nondisabling).

4. Causes of Accidents are listed as shown in Table 1.

5. Quarterly Reports are drawn up of Lost-time Injuries throughout the plant.

6. Annual Reports, signed by the Safety Director, are made in the form of tables, for: Accident Frequency and Severity, Compensation and Medical Costs, Comparison Mining Date, Comparison Accident Causes, Nature and Location of Injuries, Injuries by Months. Graphs show Frequency Rate and Severity Rate, and pertinent suggestions are summed up.

TABLE 1.—*Causes of Accidents*

Underground	Shafts	Surface Plant
1. Fall of Rocks.	17. Falling down shaft.	1. Mine cars and locomotives.
2. Handling Rock.	18. Objects falling down shaft.	2. Railroad cars and locomotives.
3. Hand Tools.	19. Breaking cables.	3. Run or fall of rock from bins, etc.
4. Explosives.	20. Overwinding.	4. Fall of Persons.
5. Haulage.	21. Cage—Skip—Bucket.	5. Stepping on nails, etc.
6a. Falling down chute, etc.	22. Other causes.	6. Hand tools.
6b. Other falls of persons.		7. Electricity.
7. Run of rock from chute.		8. Machinery.
8. Drilling.		9. Handling material.
9. Electricity.		10. Other causes.
10. Machines (Other than drills and Locomotives).		
11. Mine Fires.		
12. Natural Gas (suffocation).		
13. Inrush of water.		
14. Stepping on nails, etc.		
15. Handling material.		
16. Other causes.		

SAFETY AWARDS

Safety awards are given annually to employees who have lost no time as the result of an accident and have worked 240 shifts during the year. The awards are attractive lapel pins. The pin for one year is of bronze; for 2, 3, and 4 years, of silver; and for 5 years and more, of gold. All have the employee's name engraved on the back. The awards are given by the department heads at a public meeting.

At the time of employment each employee is given a copy of the latest edition

of the Safety First Rules and Instructions (revised each year). In each booklet is a perforated sheet containing the following form:

I hereby certify that these rules and regulations for Safety have been read, and I will do my best to uphold the same.

Kimberly, Nevada——————, 19—

Signature of Employee

Pay Roll Number————

(This Certificate must be given to Boss when Employee starts to work.)

It is assumed, therefore, that each employee knows the safety rules. However, it is imperative that constant "sales talk" of safety methods through contacts with the men in regularly scheduled meetings, instructive talks to the employees on and off the job by the Safety Director and Inspectors and members of the supervisory staff, be carried on in order to effectively impress the employee and make him "safety conscious." It has been found that only through this manner of constant effort will an average employee of a mining concern, such as the Consolidated Coppermines Corporation, fully understand the safety rules and cooperate in preventing accidents and preserving health.

Other Activities of the Safety Department

The Health and Safety Department also serves in many instances as a general service organization and participates in many of the small affairs as well as many of the celebrations and tragedies of the mining-camp community. The department has often been asked to make complete arrangements for funerals for the interment of an employee or a member of an employee's family. Fairly complete files or road maps are kept, covering most of the areas of the United States, Alaska and Canada, and a close contact is kept with many people of the community, salesman and others who visit many areas in North America, thus qualifying the department to make complete plans and cost estimates for vacation trips and to plan hunting and fishing trips to various areas for time periods covering from one to thirty days. The department has also been asked to write speeches, prepare talks and write papers for presentation by some employee at his club or fraternity. Whatever the assignment, an undivided attention is given the minutest detail, so that, as far as capabilities permit, each problem is solved as nearly satisfactorily as possible.

The Safety Department maintains an indoor rifle and pistol range and a fairly complete small "odds and ends" work and repair shop.

The rifle and pistol range is 50 ft. long and has two galleries complete with lights, backstops and bullet traps. Originally used for training the plant-protection unit during the war period, it developed into a training and recreation center for employees, Boy Scouts, State, County and City law enforcement officers, visiting peace officers, F.B.I. agents, and ex-G.I.'s, and all those interested in guns, hunting and shooting.

Rigid rules are enforced and safety instructions are given when it is deemed necessary for the safe handling of firearms. As far as can be ascertained, no person using the range has yet been involved in a shooting accident.

The workshop is available to all those who wish to repair or have repaired their broken clocks, guns, radios, sewing machines, or other necessary or unnecessary possessions. New gadgets are designed and made, ranging from small pieces of jewelry, to heavy bullet traps.

The purpose of those extra opportunities is to teach the interested employee the proper use of tools, other than the ones with which he works, in order to give him

an inkling of the duties, hazards and problems of some of his fellow workmen, and to show the Corporation's interest in his welfare and happiness in his leisure hours.

These, and innumerable other "extra-curricular courses," are given with the view of teaching the employees that the inflexible rules of safety are rigidly and impartially enforced without prejudice and for their own good; that the personnel of the department is composed of human beings with likes and dislikes similar to their own; that it is the department's duty to teach and think safety on and off the job by example as well as by word; and especially that the Corporation believes a safe man at home, on the road and among his friends is the best safety device an employer can have.

Significant Factors in Dust Control at Some Iron-ore Mines of the Lake Superior District

By Edward C. J. Urban*

(Chicago Meeting, February 1946)

The nature of certain pernicious dusts commonly encountered in the removal of iron ore from the underground mines of the Lake Superior district is recognized, and appropriate measures for the protection of individuals subjected to these dusts have been applied in most mines for more than 10 years. Undoubtedly the principal concern has been demanded by dusts that contain as a constituent silicon dioxide in its free or uncombined state, the mineral we all know as quartz or chert.

Investigators[1] agree that among the factors that determine injurious exposure are: the composition of the dust, the concentration suspended at the breathing level and the duration of the exposure. We have on hand data that describe the experience of several underground iron-ore mines in the Upper Peninsula of Michigan and Wisconsin in regard to these factors, which in a large measure also reflect the general experience of the other mines of the area. We believe that a discussion of these data will indicate the effectiveness of the dust-control methods that have been considered good practice and that have been employed over a long period of time.

A silicosis-prevention program was initiated by the iron mines of the Upper Peninsula in 1934 and records of mine dustiness are available from that date to the present time. For dust conditions previous to 1934 the results of observations conducted in other industries can be utilized together with local information pertaining to the development and initial application of rock-drilling equipment.

Development of Rock Drills

The earliest percussion rock drill that later had any industrial use was patented in this country in 1851.[2] This machine was later modified and improved in many respects but its piston-type design remained standard for drilling equipment in most mines for the next 63 years.[3] The piston drill was operated either by air or by steam. It employed a solid drill steel, fastened directly to the reciprocating piston. The drilling was done dry except in holes directed at an angle below the horizontal. For these water was manually injected into the hole and the pumping action of the rod expelled the sludge. The dry cuttings from drillings inclined above the horizontal fell from the hole by their own weight.

A development of major importance from the standpoint of dust generation and control occurred with the invention by J. G. Leyner of a hammer-type rock drill that embodied the essential feature of a drill steel with a hollow core. His patent was secured in 1897. The machine was a radical departure in drilling method. The steel was held loosely in a chuck while

Paper presented at the Sectional Meeting of the A.I.M.E. at Houghton, Mich., on June 22, 1945, and afterward revised to include additional data.

Manuscript received at the office of the Institute March 14, 1946. Issued as TP 2087 in Mining Technology, January 1947.

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[1] References are at the end of the paper.

being struck by a reciprocating hammer or piston. The hollow core provided a means whereby water, or air, or a mixture of the two could be passed through it to expel cuttings from the drill hole. Hammer drills employing a solid rod had been used to some extent but up to this time their application had been restricted to stoping or drilling upward holes. It was now possible to drill holes in any direction with the same unit. Because of the difficulty in properly fabricating drill rods with a hollow center, only a few of these machines were manufactured until a modification in the form of a hand-held jack hammer was introduced in 1912. Two years later the expiration of the Leyner patents opened the field to other builders and hammer drills rapidly superseded the piston type

Older mining men recall that hammer drills with hollow steels were first introduced into the underground mines of the Lake Superior district in 1912 and were then used primarily in rock drifting. Sinkers for shaft work and later stopers for raising, each equipped with hollow steels, followed during the next eight years. Significant developments for workings in ore were the introduction of the air-driven auger drills in 1911 and the replacing of hand mucking by mechanical methods of scraping at about 1923. The intervals of transition of drilling methods in different mines varied and were dependent on local conditions.

Dust Concentrations in Rock Drifting

In evaluating the factors of duration and degree of dust exposure associated with workings in rock, our concern is mainly with conditions that have prevailed since 1900. The 45-yr. interval from that date to the present includes the local mining experience of most of the men currently employed. It can be properly assumed (from the brief history given) that most of the drilling from 1900 to 1920 was done dry. From that time until 1934, the use of water in drilling procedures was possible but the suppression of dust was of secondary importance and the value of water as a dust-allaying device had been only partly exploited. Comparisons of dust counts during dry and wet processes in other dusty trades suggest that *the probable average dust concentrations for workings in rock during the period* 1900 *to* 1934 *were not less than* 50 *million particles per cubic foot of air.*

In 1934 control practices were applied by the local mining industry, designed specifically to reduce the dust hazard in working environments. The essential requisites for workings in *ore* were planned methods of mine ventilation to provide each work place with an adequate amount of fresh air, wet drilling where practicable, the rapid removal of contaminated air from workings and common sense dust control practices to be followed by the miners themselves. For operations in *rock*, emphasis was placed on wet drilling, the copious use of water for spraying down the surfaces of the work place and the muck pile during loading and the provision of forced ventilation to the heading. The use of air-line respirators by the rock miners was an important adjunct, particularly where ventilation was not entirely satisfactory.

Fig. 1 reveals the combined or average dust concentrations during rock workings in eight underground iron-ore mines of this district through the 10-yr. period from 1934 to 1943. The operations involved were those common to main drifting and included wet drilling, scraping or loading muck, timbering and other work common to such development. The effectiveness of the methods of dust control employed in each of these eight mines was comparable to the average for the region as a whole. This particular group of mines was selected simply because for these mines the desired information was available.

The trend indicated on the chart by the irregular solid line was developed by averaging all the dust counts made at the eight mines for four-month intervals and problem of smoke and dust in the mine work place. The immediate effect was a distinct trend toward larger volumes of mine air and auxiliary ventilation systems

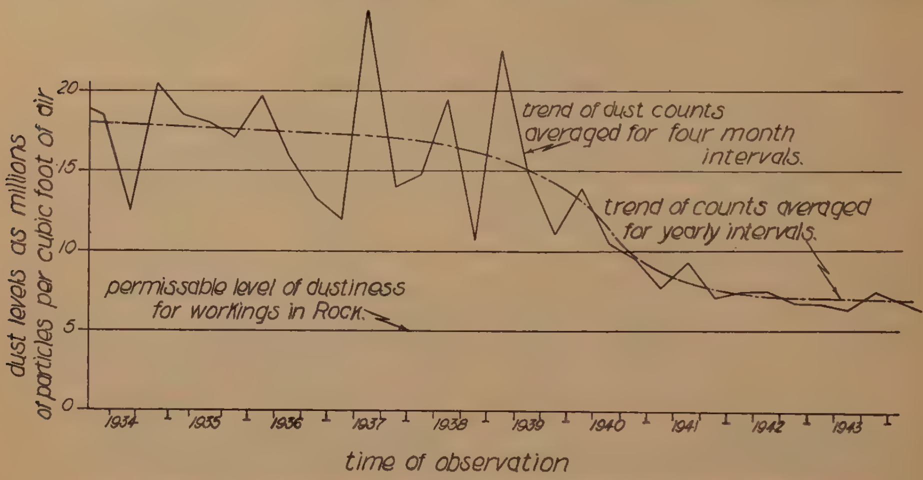

FIG. 1.—TREND OF AVERAGE DUST CONCENTRATIONS IN ROCK DRIFTING UNDERGROUND. This includes scraping or loading muck, drilling wet, timbering and general and is based upon the combined experience of eight different iron-ore mines (1200 air samples during the period 1934 to 1943).

plotting the magnitude of each result in millions of particles per cubic foot of air as ordinates. The period corresponding to each value represented was plotted as abscissa. The smooth curve indicated by the broken line was developed in a similar manner except that the same dust counts were averaged for intervals of one year. The chart shows that levels of dustiness during the early years of the control program varied within limits but averaged 18 million particles per cubic foot in 1934. The general level decreased to 15 million in the next five years and then dropped abruptly to about 7 million in 1941, and that level was maintained through the remaining period of observation. It should be emphasized that previous to 1939, when higher counts prevailed, the common use of air-line respirators afforded rock men adequate protection.

Increased mining activity in the period following 1939 greatly augmented the of higher capacity, and added attention to the control of air-borne contaminants.[4] The resulting improvement in air conditions underground reduced the need for personal respiratory protective devices to a minimum. The picture of dust concentrations varied in different mines, but in general it followed a similar pattern. In some of the mines where greater attention had been given to details of ventilation, dust levels in rock approximated 5 million particles per cubic foot (the present standard of permissible dustiness for *rock*) prior to 1941 and subsequently have been maintained.

DUST IN ORE WORKINGS

The available data are not in proper form at this time to allow a similar analysis of dust conditions associated with workings in ore. Most merchantable ores are defined in part by a low silica content and it can be properly concluded that the silica

hazard involved in exposure to moderate amounts of dust from such material is essentially small. Unquestionably, conditions of excessive dustiness in ore workings were not uncommon previous to 1934. In fact, for some time after this date, high dust counts were often recorded, particularly in raising. More recently considerable attention has been given to atmospheric control in ore workings. The introduction of larger volumes of air now eliminates excessive dustiness, rapidly removes smoke from the mining area, prevents the possibility of oxygen deficiency and generally improves working conditions.

Dust concentrations in ore tend to vary more than those in rock because they are influenced largely by the nature and the degree of wetness of the ore, the type of mine-ventilating scheme employed and contamination from such adjacent operations as blasting. A number of mines have now established 5 million particles per cubic foot of air as the permissible level of dustiness for *ore* as well as rock workings. In other mines the average dust levels approach 10 million particles per cubic foot.

Composition of Rock Dusts

The composition of the dusts comprising the exposures under discussion can be described by considering the nature of their source. The rocks commonly encountered in mining on the Marquette Range can be generally classed as: Goodrich quartzite, Negaunee iron formation, Siamo slate and Ajibik quartzite. On the Menominee Range they are: graywacke, gray and black footwall slate, iron formation and graphite black hanging-wall slates. These common to the Gogebic Range are: quartzite and quartz slates, the Ironwood iron formation with its cherty members and the interbedded Tyler slates. These different rocks vary in their mineralogical characteristics but they are all of sedimentary origin and all contain considerable amounts of silicon dioxide in the form of quartz or chert. In addition, igneous intrusives such as greenstones are frequent on the Marquette Range, and such acid and basic intrusives as diorite occur on the Gogebic Range. Greenstone and granite underlie the iron formation on the Gogebic.

The common practice in all mines for several years has been to evaluate the dustiness of atmospheric samples by the standard U.S.P.H.S. light-field technic and report the counts as "total dust particles less than 10 microns in size per cubic foot of air." For added information, most mines have sampled the rock being mined at the same time that air samples were being collected for counts. These rock samples were analyzed for *total silica* content. The results, based upon rock, were used in estimating the degree of siliceous hazard from the air-borne dusts. The simpler analysis for total silica reveals the entire amount of silicon dioxide present, both that in the free form as quartz and that combined as silicate. Since only *free silica* has much hygienic significance, it is particularly desirable to establish the proportion of such silica in the rock. This is most conveniently done by X-ray diffraction.

Results of Analyses

In all, 527 samples of various rocks were collected from different headings in five mines of the Marquette Range during the interval from 1934 to 1943; 165 were similarly obtained from four mines of the Menominee Range and 586 from five mines on the Gogebic. All are presented in Fig. 2.

The percentage of *total silica* was used as the abscissa and the cumulated frequency of the samples weighted in terms of the percentage of the total number for each series was plotted as ordinate. The curves may be interpreted as follows:

one sixth of the samples from the Marquette Range showed up to 28 per cent silica and five sixths of the same series had 58 per cent or less silica. The analyses the nature of the silica in the iron formation on the Menominee was similar to that of the iron formation of the Gogebic. The method of reporting analyses on the

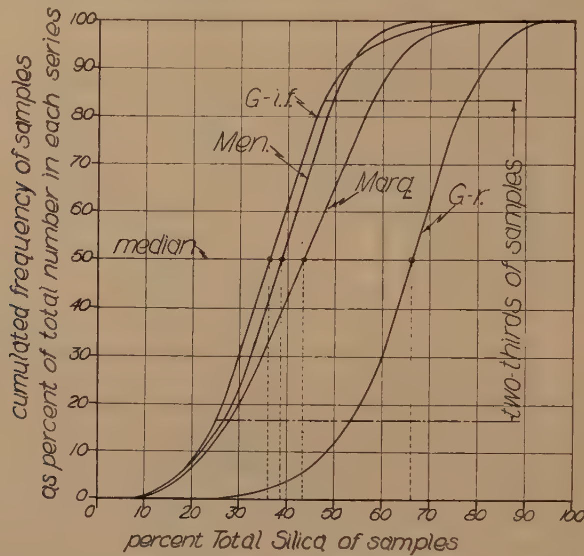

FIG. 2.—FREQUENCY DISTRIBUTION OF THE TOTAL SILICA CONTENT OF ROCKS ENCOUNTERED IN MAIN HEADINGS UNDERGROUND OBSERVED DURING PERIOD 1934 TO 1943.
G-i.f. Gogebic Range, iron formation, 409 samples.
G-r. Gogebic Range, quartzite and quartz slate, 177 samples.
Men. Menominee Range, iron formation and other rocks, 165 samples.
Marq. Marquette Range, iron formation and other rocks, 527 samples.

from the Gogebic Range had been classified in more detail as to kind of rock. The two curves shown in Fig. 2 relate to the rocks most generally encountered in mining on that range; one for iron formation (*G-i.f.*) only and the other for quartzite and quartz slate (*G-r*). It was interesting to note in the compilation of these data that the frequency distribution of the silica content of the same kinds of rock on the Gogebic was essentially the same in whatever mine it occurred. The curve shown for the Menominee is a composite of the different rocks common to that range; the frequency distributions of the analyses of the graywacke, the black and gray slates were generally similar and showed as only slightly more siliceous than the iron formation. In addition, Marquette Range allowed for but a single composite curve of the different rocks. It does show, however, that one half of the samples had a silica content of more than 44 per cent. Undoubtedly, these were the quartzites and slates.

It can be properly assumed that the silica hazard in rock mining in the Lake Superior district varies with the nature of the specific rock encountered. The limits of this variation are indicated in Fig. 2 by curves *G-i.f.* and *G-r.* The former represents the common iron formation, which has the lowest silica content, and the latter, the highly silicious quartzites and quartz slates.

In an attempt to establish a relationship between the proportion of *free* and *total* silica in air-borne dusts, we have analyzed

TABLE 1.—*Comparison of Free and Total Silica Content of Rock Dusts Collected from Dry Pneumatic Drillings*

PER CENT

SAMPLES FROM MARQUETTE RANGE	
Siamo slate and graywacke:	
Total silica	53.0
Free silica	48.8
Siamo slate:	
Total silica	54.8
Free silica	29.7
Total silica	52.2
Free silica	34.8
Ferruginous chert or iron formation:	
Total silica	40.1
Free silica	36.4
Total silica	47.6
Free silica	41.4
SAMPLES FROM MENOMINEE RANGE	
Gray slate:	
Total silica	39.7
Free silica	35.0
Cherty iron carbonate:	
Total silica	40.16
Free silica	36.7
Black graphitic slate and chert:	
Total silica	35.5
Free silica	29.5

dust and chips collected directly from dry pneumatic drills. The collecting apparatus included a dust trap designed to enclose the drill steel while drilling dry, a suction fan and suitable filter. The cuttings from four 3-ft. holes usually were sufficient to yield about ½ cu. ft. of material. The percentages of free and combined silica for eight such samples are shown on Table 1. The number of comparisons presented is small, but they indicate that in most of the sedimentary rocks encountered in these mines the silicon dioxide is largely in a free state. The greatest difference between total and free silica value is noted for the Siamo slates, in which the quartz constitutes only slightly more than half of the total silica. Both samples came from the same mine but the individual variation may be due to faulty selection of materials.

SUMMARY

In the foregoing presentation we have discussed factors that are significant in the application of the dust-control program now in effect in the iron-ore mines of the Lake Superior district. The purpose of this program is to prevent the development of new silicosis and to protect from further progression of their condition old miners who have already developed some dust reaction from previous exposure.

The current permissible levels of dust concentration *in ore* workings are not maintained essentially for the control of silicosis but to ensure efficient general mine ventilation. For workings *in rock* the specific purpose of the standard of 5 million particles of dust per cubic foot of air is to reduce the silica hazard to a minimum. This standard has been developed as a result of medical and engineering experience and is in accord with similar standards maintained in other dusty trades.

The dust level trends reviewed do not represent the experience of any one mine; they form the composite picture of data from 35 different mines.

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Mining by Top Slicing at the Negaunee Mine, Michigan

By W. R. Atkins * Member A.I.M.E.

(Chicago Meeting, February 1946)

The Negaunee mine is at the east end of the Marquette Range, in the city of Negaunee, on the Upper Peninsula of Michigan. Iron ore was first discovered on this property in 1883 by diamond drilling. A shaft was sunk soon afterward, and the mine has been worked continuously ever since. The total shipments from 1887 to Jan. 1, 1945, were 20,417,587 long tons. The mine was taken over by the present operators, The Cleveland-Cliffs Iron Co., in 1903. During the past four years, 1941 to 1944 inclusive, the property has produced an average of nearly a million tons a year, and has employed more than 400 men.

Geology

The ore body at the Negaunee mine lies in a large syncline dipping about 15° to the west, the ore coming to ledge at the east end of the trough. The ore occurs in the Negaunee formation of the middle Huronian age, which consists of ferruginous cherts and slates through which intrusions of diorite have occurred. Although these intrusions of diorite have an important bearing on the formation of ore on the range, they are not important in this ore body. Some faulting has taken place along the diorite intrusions, or dikes as they are called locally, and also other faults are known to exist across the formation. These faults do not complicate either mining or development of the ore body. The footwall consists of jasper underlain with Siamo slate; the hanging wall is jasper and ferruginous chert. Fig. 1 shows a typical cross section of the ore body.

The ore body is quite regular, about 120 ft. thick in the middle of the syncline and thinning out toward both sides of the fold. Its width ranges from 400 to 800 ft., the average being about 600 ft. The footwall is uniform in dip and strike, but irregular lenses of ore often extend 50 to 75 ft. up into the overlying jasper capping.

Method of Mining

Ore in the Negaunee mine has been mined by top slicing almost exclusively since 1903. This method of mining has been employed because it is especially adapted to the size and shape of the Negaunee ore body, and to the soft, sticky nature of the ore as well as the generally soft, broken jasper capping.

Development

The ore body has been developed from six different levels with intervals varying between 100 and 125 ft. On the average, the dip causes a horizontal offset of 350 ft. per level, and under these circumstances it has been necessary to carry on considerable development in rock to make available the ore lying along the north and east footwalls. This is not as disadvantageous as might be supposed, because of reduced pressure and the accompanying saving in maintenance costs. The development of the

Manuscript received at the office of the Institute April 24, 1946. Issued as TP 2091 in Mining Technology, November 1946.

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FIG. 1.—TYPICAL CROSS SECTION OF NEGAUNEE ORE BODY, SHOWING DIORITE DIKE INTRUSIONS.

two most recent levels, the 13th and 14th, was carried on before 1940. The main untimbered shaft drift is driven in slate and graywacke to a point approximately 200 ft. away from the ore contact, and here the drift branches out into six different timbered crosscuts, which are driven under and into the ore. The general practice has been to drive the crosscuts, whenever possible, at right angles to the strike of the ore body or through the bedding planes.

Where ground pressure is evident, crosscuts in this position seem to withstand the squeezing pressure considerably better than drifts running with the formation, other things being equal.

During the past three years, because of the heavy mining schedule and the rapid depletion of segregated mining areas, mining by top slicing has been concentrated for the most part on one main sublevel in an area between the footwall and jasper capping approximately 400 by 600 ft. This area is served by six 14th level crosscuts, from which raises are extended to the active sublevels. Two-compartment cribbed raises, each compartment 4 ft. 2 in. square inside, are put up at an inclination of 68°, one compartment serving for the storage of mined ore prior to loading in tramcars and the other for a manway. The distance between raises varies with the ore body, and is dependent on the area or block to be mined. A plan map of the active mining sublevel together with the main level development is shown in Fig. 2.

As indicated on the sublevel map, mining operations radiate from the various raises and each raise and mining area is occupied by a gang or contract. Frequently a mining contract will remain in the same area, mining on successive 12-ft. sublevels from the jasper capping down to the lean-ore footwall, a distance comprising possibly 120 feet.

The contract miners are paid in proportion to the number of 4-ton tramcars per 8-hr. shift (collar to collar). During periods of timbering or nonproductive work, a fixed hourly rate is paid, which is averaged with the contract rate for each two-week pay period.

In development on and above the main levels, all rock and ore drifts and raises are paid for on the footage basis. This rate varies with the hardness of the ground, size of opening and other conditions that may interfere with normal progress.

In all main-level drifts serving as permanent haulage or ventilation ways and requiring support, treated timber is used. The preserving agent is zinc chloride, and the treatment extends the life of the timber from an average of 3 years to well over 12 years. A considerable saving is effected in costly repairs and maintenance.

Top Slicing

Top slicing as practiced at the Negaunee mine is a method of mining ore by a series of 10 to 13-ft. contiguous horizontal layers called sublevels. The progress of mining such layers is from the top down, and the ore is handled through timbered raises. At this mine, a layer of ore is extracted by a series of timbered slices arranged radially from a raise to cover approximately equal areas between raises. As a slice is advanced, the space resulting from extracting each cut of ore is timbered—a set for each cut. After a slice is mined and the space is timbered, the floor is covered with poles, lagging or slabs, and the sets are blasted down. When the back of a slice has caved, adjacent slices can be mined with comparatively safety. The successive layers of poles and blasted mining timber provide a mat, which acts as a cushion and distributes the weight of the overburden. Mining by this method of top slicing is adapted to soft ore deposits of large lateral extent, and yields a high

percentage of extraction with a minimum of ore dilution.

Top slicing was first carried on in Lancashire, England, and was adopted and only a short time afterward the double-drum electric hoist, powered to scrapers, was put into use. This was a great forward step in mining efficiency.

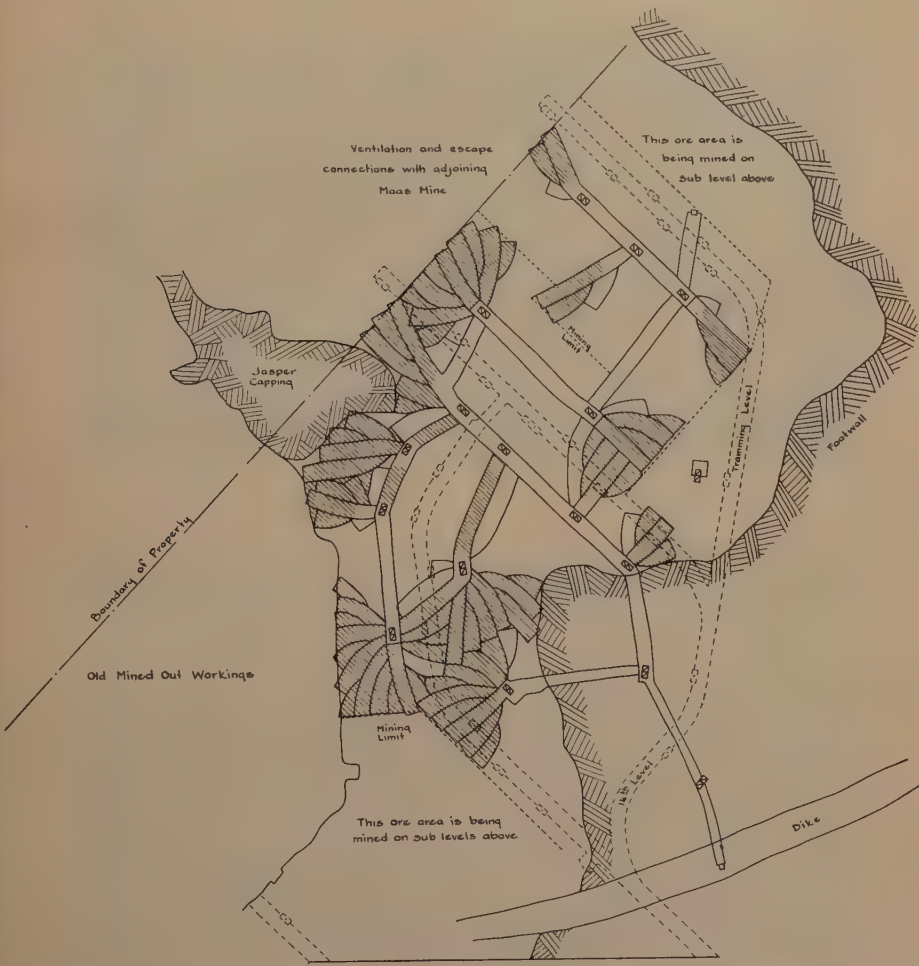

FIG. 2.—SUBLEVEL PLAN MAP SHOWING METHOD OF DEVELOPMENT AND ARRANGEMENT OF SLICES.

in the Michigan district because of the soft nature of the ore. Before the introduction of the scraper hoist, the moving of ore in sublevel drifting and slicing was carried on by the use of small, one-ton cars, or "buggies." The cars were filled by hand shoveling at the breast or face and trammed to the raises. Later, several types of air-operated loaders were tried,

A little before that time a method of inclined top slicing was started, where the slices were driven at an angle with the horizontal to facilitate moving the ore to the raise, but this did not prove advantageous because of the difficulty and hazards involved.

Prior to 1930, mining by top slicing was carried on by using a parallel slicing

method in each horizontal layer or sublevel. In the original development, the raises were spaced in the various crosscuts at 35-ft. centers, and a mining contract on the sublevel drifted to the mining limit at right angles to the line of raises. On either side of this drift two slices were driven. A similar operation was performed in the opposite direction from the raise. The mining contract then moved to an adjacent raise, and the operation was repeated. This method did not prove entirely satisfactory, owing to the difficulty of recovering small pillars and because of the large amount of development and accompanying high maintenance cost.

During the past 15 years the top-slicing mining method has been changed from the parallel type of slices to the radial type. This lessens the amount of development required and allows a larger sublevel area for each mining contract. The slices radiate from the raise location and the resultant curves do not greatly affect the scraping of the broken ore.

Mining Cycle at Negaunee

The sublevel drifts are all timbered, using sets with 9-ft. legs and caps at 5-ft. centers. In advancing a drift from the raise preparatory to slicing, approximately nineteen to twenty-three 5½-ft. holes are drilled, using a standard jackhammer and the conventional auger-type twist drills (Fig. 3). The blasting of an average cut requires approximately 90 sticks of 1¼-in., 40 per cent powder. All holes are tamped with two 1½ by 8-in. shells, which are filled with the fine ore from the drill holes. In recent years particular attention has been given to the blasting practice, from both a safety and an efficiency standpoint. The fuse is lit by use of a hot wire lighter, which burns for approximately 60 sec., and all miners are instructed to leave the working face when the lighter has burned to completion. After the blast the pile is partly leveled off, to allow the back to be trimmed to the mat. Four 5-in. tamarack forepoles are put in over the cap to cover the back or matting of the sublevel above. This operation is illustrated in Fig. 4. A covering of cedar lagging or slabs is then laid across the poles, to protect the miners from material falling from the mat while they are scraping out the blasted ore and putting up the next set. Three gin poles are installed between the breast and the last set, high enough to permit

Fig. 3.—Looking toward the breast, showing a miner using a drill machine mounted on a jackleg while his partner completes the spragging of the last timber set.

the next timber set to be installed without removal of the poles. These poles carry the 8-in. head block, used in scraping the broken ore and to aid in holding loose

FIG. 4.—MINERS INSTALLING FOREPOLES THAT EXTEND UNDER EXPOSED COVERING OR MAT.

FIG. 5.—AFTER THE BLAST. SUFFICIENT ORE IS BEING SCRAPED TO ALLOW ROOM FOR FOREPOLES. NOTE OLD BLASTED TIMBER ON RIGHT SIDE OF SLICE.

ground immediately under the timber mat. As an added precaution, in all drifts advancing under new hanging wall or where ground is left in the back, two 12-ft. 4-in. H-beams are used alongside the forepoles.

The scraping operation requires one to two hours, depending on the length of the slice and the amount of storage available in the ore compartment of the raises. Fig. 5 shows a typical scraper in operation.

Of the four main parts of the mining cycle, drilling, blasting, scraping and timbering, the latter requires the most time and skill. Frequently timbering and scraping are carried on in conjunction. Experienced miners can complete this operation in about two hours, and often one man commences to drill the next round while the other completes the final

bracing. During the drilling or timbering operation a three-man timber-hoisting crew, going from one contract to another, will hoist sufficient timber to allow the contract at least one set in advance of the operation. Toward the end of a mining sublevel, some delay is caused by lack of storage room as well as low head room in the vicinity of the raise. The complete cycle, including drilling, blasting, scraping and timbering, requires between five and seven hours, depending on general conditions.

Where mining is carried on under the crushed jasper capping, sometimes rock runs occur after blasting, which greatly retard normal progress. In this case it is necessary to drive pointed tamarack poles over the last cap across the width of the slice. The best results are obtained by allowing the opening to remain filled. If the rock is scraped, larger chunks prevent the driving of spiling and greatly hamper the operation. In recent years, the pushing of poles through the broken rock has been accomplished by the scraper hoist, and to facilitate this work a 5-in. pipe with a small sheave attached to the back, called a "Glory-be," is used as shown in Fig. 6. This device doubles the power pull and greatly speeds up the work. The causes of rock runs are: broken jasper capping, poor or insufficient covering, drilling the back holes of the blast too high, and the leaving of small, uncovered areas on the sublevel above.

Fig. 6.—Use of the "Glory-be" in pushing spiling ahead of last set to prevent a rock run or old wood in back from dropping down.

On the completion of a slice or sublevel drift, considerable attention is given to the covering on the floor. Poles 9½ ft. long and 6 in. in diameter are laid along the drift and spiked to three cross pieces. Under new hanging wall near dikes or horses of jasper, or where loose rock has run during the progress of mining, these poles are placed close together (Fig. 7), or heavy wire netting is laid on them to protect the next sublevel from runs of rock. Double covering poles are also used at the end of a drift and at the limit, where mining must be carried on adjacent to this area. In wet places, or where there is an accumulation of bottom water, it is often necessary to lay poles on the bottom of the drift as it is advanced. In some places, where the drift will drain the adjoining mining area, a 2-in. hardwood plank floor

is laid, and the water is piped to the raise. This affords an excellent covering for the sublevel below, particularly if similar wet conditions are experienced.

in a fireproof building at the collar of an idle shaft at the east end of the property. The building also houses a steam plant, which is used during severe winter weather

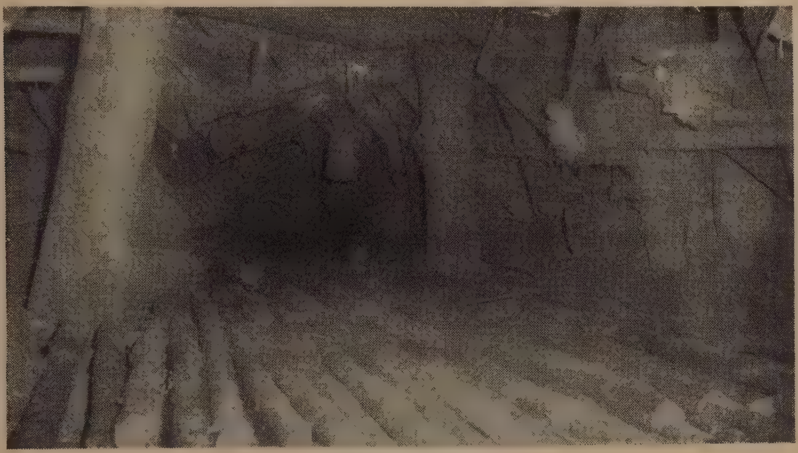

FIG. 7.—COMPLETED SLICE.
Showing close covering poles on floor as used under loose rock or broken jasper capping. Note that inner portion of slice as shown in background has been blasted down.

to raise the temperature of the air sufficiently to prevent the icing of the shaft. This heating plant is equipped with 12 large unit heaters, which operate automatically with the temperature of the incoming air.

The ventilating shaft connects with the 9th level. One small split ventilates five mining contracts above the level. The remaining air is then directed to the 12th level through large rock raises, where there is another split of 15,000 c.f.m. to the adjoining Maas mine. About 2000 c.f.m. is allowed to leak through air-lock doors on each level, to ventilate drifts and stations at the hoisting shaft. The remaining air is carried to the 13th level, where there are five splits to distribute air to active mining areas between the 13th and 14th levels. Each split ventilates five or six contracts. During the past two years, 22 contracts have been operating in this area, and it was found that those near the center of the ore body did not receive sufficient air. To correct this

On the completion of mining operations in the vicinity of a raise, and before the raise is cut for a new sublevel, two 12-ft. hardwood sills are laid over the raise near the hanging-wall side. From these sills props are set up to support the timber over the raise, then the miners cut out the raise at a point approximately 13 ft. below.

The setting of mine limits varies considerably on each sublevel, in order to prevent a line of weakness from being established in the timber mat. These limits are not always located parallel to the line of raises or in any set pattern. Their location depends on the speed of mining, length of scraper haul, and dikes or jasper inclusions that may bound or intersect the ore body.

VENTILATION

Ventilation at the Negaunee mine is provided by a Jeffrey Aerodyne fan of 125,000 cu. ft. per min. capacity with an auxiliary American blower fan of 100,000 cu. ft. per min. capacity. The fans are

condition an auxiliary fan of 17,000 c.f.m. capacity was installed on the 14th level in the third compartment of an idle raise. This fan directs the air into this inner a short raise equipped with fog nozzles to the adjoining Maas mine.

As mining continues on a sublevel it is frequently necessary to reroute the air

FIG. 8.—COMPOSITE MAP OF TWO MAIN LEVELS SHOWING VENTILATION SYSTEM OF MINING AREA.

area on the main sublevel, where it is distributed to the mining contracts. Two ¼-in. water-fog nozzles are installed in the raise to cool and clean this recirculated air.

In an effort to increase the circulation of air in the 1420 and 1430 crosscuts, which connect on the 14th level as shown in Fig. 8, a second 17,000-c.f.m. booster fan has been placed at the extreme southwest end of the drift, which exhausts up because of crushed ventilation drifts. It has been found more economical to drive new traveling and ventilation drifts beside the crushed drifts than to maintain the original ones. On the 14th level there are three connections with the adjoining Maas mine, at the ends of the crosscuts. They are equipped with water curtains, which clean and cool the air and retard the decay of timber in the drifts beyond. In dead ends and in areas where the sub-

level is nearing completion, auxiliary booster fans are installed near the raises on the level. Rubber-covered 12-in. canvas tubing delivers the air to the sublevel. Under normal conditions approximately 2000 to 3500 cu. ft. of air pass through each active contract on the large mining sublevel. Ten to fifteen minutes is required for blasting smoke to clear.

Equipment and Production

At the time the mines in this district were opened, and later as top slicing came into practice, the total equipment of a mining contract amounted to the old type jumper and drill, a No. 2 shovel, a pick, a small buggy and an assortment of 12-lb. rail. Probably the total outlay amounted to less than $200. At the present time the cost of equipping a mining contract amounts to roughly $2200. This cost does not include the equipment and expense involved in servicing and maintaining these tools.

During the war period, in 1942, the production at the Negaunee mine amounted to 1,106,700 long tons of ore. This tonnage required one 9-ft. timber for each 14 tons of ore. The total amount used was equivalent to 263 railroad cars of timber together with 150 cars of lagging and 126 cars of poles. This tremendous amount of timber is rapidly depleting the forests of the Upper Peninsula, and just how long this supply will last is a matter of considerable concern.

During 1942, 280 tons of powder was required—approximately ½ lb. of powder per ton of ore.

The average cost of labor in 1916 amounted to approximately 64 per cent of the total cost of production; in 1945, it was 54 per cent; that is, inversely proportional to the increased use of mechanical equipment.

The average product based on the total number of employees at the Negaunee mine in 1916 was 4.99 tons per man per day as compared with 7.62 tons in 1945. This increase can be attributed to possibly four factors: (1) improved mechanical equipment; (2) better working conditions, particularly with respect to ventilation; (3) greater attention and education in safety practices; and (4) a general improvement in the planning and carrying out of the top-slicing method itself.

Fire Hazards and Safety Work

With the tremendous amount of timber used in top slicing there is always considerable danger from fire, and very definite preventive steps have been taken at the Negaunee mine. All raises, drifts and escapeways are clearly marked to allow rapid evacuation from all working places in the mine. The three main exits to surface include the ventilation shaft, the main hoisting shaft and the adjoining Maas mine, to which numerous connections are maintained.

A two-man fire-inspection crew visits each working place in the mine shortly after the last shift in each week and once every 24 hr. thereafter during idle periods. All electric power switches are checked, and a report is made as to conditions found by this crew.

In addition to a large number of fire extinguishers, which are checked periodically, there is a tank and hose car on each active level. Each car has a 100-gal. tank of water, to which an air connection can be made, and a 200-ft. hose on a reel, which can easily be carried up a raise. These tanks are fully charged and ready for use at all times. A 2-in. water line is maintained on all active levels with a pressure in excess of 60 lb., to further facilitate the fighting of fires.

Smoking is prohibited in the mine, in the timber tunnel to the shaft and in the timber yard. Any violation of this rule leads to immediate discharge.

Stench Warnings

A stench warning device is on the main air line in the hoist room, charged with a 1000 c.c. cartridge of ethyl-mercaptan, for use in emergency. The hoist engineers and floormen have been instructed in its use. Underground employees have been taught what to do if the stench warning odor is detected underground. Yearly tests are made with this method of warning, while men are on regular shift.

Mine Rescue

Twelve men are trained each year in the use of self-contained oxygen breathing apparatus and all-service gas masks. These men are available for mine rescue and fire fighting. Besides these men, 121 men employed in near-by company mines have had this same training, and are available in case of emergency.

First Aid Training

First-aid training is available once each year, and is conducted either by the company's instructors or the U. S. Bureau of Mines. Approximately 150 men have now had this training at the Negaunee mine.

Diamond Drill Blasthole Stoping at the Book Mine, Menominee Range, Michigan—Progress Report

By L. S. Chabot, Jr.,* Member AIME

(New York Meeting, February 1948)

Introduction

The bibliography of mining methods in the past few years has contained many articles dealing with the use of the diamond drill for blasthole drilling.

In the Canadian mines, this method has been used with great success in stoping at Noranda, Aldermac, Waite Amulet, East Malartic and Sladen Malartic mines in the Province of Quebec. It has also been used at the Dorchester and Beattie gold mines in Duparquet, Quebec. The method has found useful application at Copper Mountain mine, British Colombia, and at Flin Flon mine in Manitoba.

In the United States, the diamond drill blasthole method has been used at Cornwall Iron Ore Mines in Pennsylvania and at the Republic Steel Corporation magnetite properties in the Port Henry district of northern New York.

In the Lake Superior iron district, the method has been used at the Anvil-Palms mine of Pickands, Mather and Co. at Bessemer, Mich., and is also used at the Book mine of the North Range Mining Co. at Alpha, Mich. Other companies in the district have also experimented in some degree with the method at some of their properties.

This paper is being offered as a preliminary report on the use of the diamond drill method of stoping at the Book mine. Sufficient work has been done with it to prove its worth and it is the purpose of the article to present some results, operational details, equipment ideas and recommendations which we have found in its use at our mine.

Location and Description

The Book mine is situated about a mile north of the village of Alpha, in the NE-NE¼ and the SE-NE¼ of sec 12-42-33, between the Dunn mine and the Balkan-Judson mine, and about four miles southwest of the village of Crystal Falls, on the Menominee Range of Michigan.

The property was found to have ore near surface in sufficient quantities for open-pit operations and stripping was begun in the latter part of 1942 to prepare it for ore production in 1943. During 1943, 288,447 tons of direct shipping ore were produced from the open pit and 382,933 tons in 1944.

While the mine was being operated as an open pit some 350,000 tons of lean ore were stockpiled for later treatment. This material is being treated in a washing and jig plant and concentrates are produced during the shipping season.

The ore bodies at the Book mine are characteristic of the Menominee Range. They are found on the flanks of sharply folded anticlines which have a fairly steep pitch to the northwest. The open-pit operations in 1943 and 1944 revealed two of these anticlinal structures. A detailed study and classification of the old diamond drill records, coupled with the information of some new holes drilled, has established the

Manuscript received at the office of the Institute Feb. 21, 1947. The paper was presented at the 20th annual meeting of the Minnesota Section, AIME, Duluth, Minn., Jan. 13, 1947. Issued as TP 2305 in Mining Technology, January 1948.

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presence of four additional anticlines with ore possibilities.

The operations in 1944 exposed ore along the west wall of the open pit which was not available by open-pit methods because of the adverse stripping ratio. At the close of the shipping season that year, an inclined skip way was installed on the slope of the east wall of the pit, at an angle of 26°.

The skip way, approximately 700 ft long, is double-tracked for half its length to accommodate two skips in balance. About 50 ft below the midpoint, the two tracks converge through a frog into a single track system, with double lead rails, for the remainder of the distance down into the pit where it passes under a loading pocket.

An adit has been driven into the west wall of the pit from this loading pocket and mining operations are in progress by underground methods. Drifts are turned off to the north and south from the adit, or crosscut, into the mine and stoping operations are carried on under the west wall of the pit to recover the ore remaining in the flank of a large anticline at and above the elevation of the pit bottom.

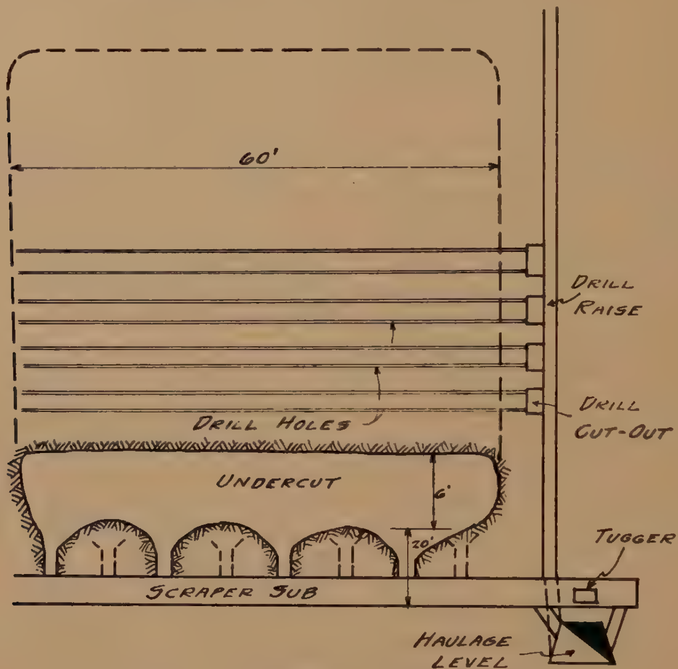

FIG 1—LONGITUDINAL SECTION OF STOPE.

The skip way at surface extends over a loading pocket from which railroad cars are filled during the shipping season.

Stockpiling of ore during the winter months is done with Koehring dumptors which had been used for ore haulage during the open-pit operations. These dumptors are used in the summer months for transporting lean ore from the lean ore pile to the wash and jig plant for beneficiation.

Mining Methods

Sublevel stoping by conventional methods and diamond drill blasthole stoping are the two methods used for the production of ore.

In the sublevel stoping, the scraper-subs are established a few feet above the back of the main haulage drift, with a short chute

for loading the tram cars. The scraper-subs are sometimes at right angles to the line of the main drift and in some cases parallel the main drift, depending on the attitude of the ore bodies being mined. In some instances, the stopes have been broken through to the open pit and a good view of these underground stopes can be had from the banks of the open pit.

Mill raises are brought up from the scraping-sub a distance of 20 ft above floor elevation. The mills are coned out at the elevation of the mill sub and cut through from one to the other at the far end of the stope block for the beginning of the slot, which is carried up to the full height of the stope block before the face is retreated. Access to the slot is had by additional sub-level dog drifts driven at 20-ft intervals from a travel-way raise at the entry to the stope block.

In addition to the sublevel stoping method, some areas of the mine have been developed for stoping with the use of the diamond drill for blasthole drilling (Fig 1 and 2).

These stopes are laid out with a width of approximately 60 ft, and a pillar approximately 25 ft wide is left between adjacent stopes. The length of the block to be drilled with the diamond drill is about 60 ft.

The stope block is developed in the following manner. The scraper-sub drift is driven a few feet above the back of the main level drift, as in the sublevel method, mills are brought up from each side of the scraper drift and are coned out at a height of 20 ft above the floor of the scraping sub. The stope block is then undercut for the full width across the far end, and the rib of the undercut is worked back at a height of approximately 6 ft above the mill sub elevation, until the entire length of the proposed block is reached.

The method used in the blasthole diamond drilling is the so-called, "Horizontal-Radial" system and is applied to the stope block, which has been undercut, in the following manner.

A small raise is driven from the scraper-sub to the top of the ore and about 10 ft

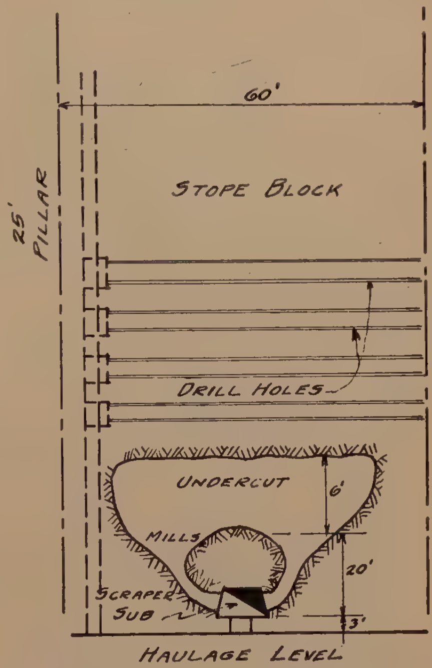

Fig 2—Cross section of stope.

beyond the proposed limit of the stope. This raise is planned so as to come near a corner of the stope and on the line of the pillar. Cut-outs are made in the raise about 12 to 15 ft apart, floor to floor, for setting up the diamond drill, which is mounted on a standard drill column having a double jack screw to ensure rigidity. The cut-outs are sometimes staggered with relation to one another if the ground is weak, but are taken in the direction of the stope (Fig 3).

The position of the raise allows the drilling of a radial pattern of horizontal holes, starting with the first, along the line of the pillar, to the last hole, which is roughly across the width of the stope block.

The holes are turned off with a large metal protractor on a predetermined angle

so as to give a spacing of 6 ft at the ends of two adjacent holes.

The cut-outs for the diamond drill are high enough to permit drilling one round of holes at the bottom of the column and another about 8 ft above the lower round, near the top of the column. Staging is placed in the raise to facilitate drilling the upper round. The first cut-out is accurately measured so as to give a burden of 8 ft over the back of the undercut which has previously been made over the entire stope block.

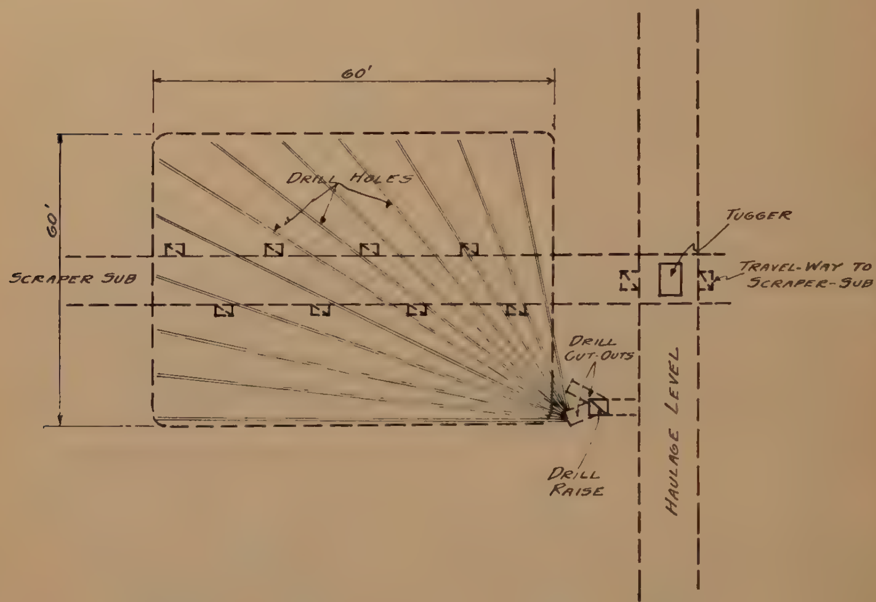

Fig 3—Plan view of stope.

Equipment

The diamond drill used is a Sullivan HS-15 air-driven, screw-feed type. The feed speeds are 200, 300, 450 and 600 revolutions per inch of advance and the drill itself has an 1800 and 3500 machine rpm.

In the type of ore which we are drilling we find that a feed speed of 300 revolutions per inch and a machine speed of 3500 rpm is the most suitable for our operation. The 1800 machine rpm is recommended when breaking in a new drill.

The *EX* size of bit and rods are in use. The rods are in 2-ft lengths and a short core-barrel, 5 ft or less in length is an advantage because of the small space in the raise cut-out. The use of a core-shell set with bortz strips improves the life of the drill bit.

This report covers the use of three types of bits; the concave, non-coring; the pilot, non-coring; and the standard *EX* coring bit. All the bits were bortz set in a hard alloy metal matrix and were not all of the same manufacture. In the period of this progress report 31 bits were used.

Drilling

Water for the diamond drill is fed from a small Goulds triplex pump placed on the main level. A pressure of 100 psi is maintained at the drill and the pump now in use can serve two diamond drills.

The first hole is carefully surveyed and

located for direction and proper burden over the undercut. The succeeding holes are turned off and drilled according to a plan furnished by the mining engineers which gives the length of each hole and the relative angle to the adjacent hole. It must be understood that all the holes are not of the same length because the drill raise is in a corner of the stope block and the hole across the diagonal of that block is the longest hole. All holes are drilled so as to provide a space of approximately 6 ft between each hole at its far end.

The diamond drill footage in this preliminary report was 2650 ft.

Blasting

In our experience at the Book mine, a great deal of care was required in the loading of the diamond drill holes because the whipping action of the rods enlarged some of the holes at points along the length of the hole.

When a hole has been drilled in fairly hard ground, it is smooth and has not enlarged much beyond the diameter of the drill bit. Such a hole is charged in the following manner. Ensign-Bickford Primacord is fastened to an 1⅛ by 8-in. dynamite cartridge, which fits into an *EX* drill hole, and is pushed to the bottom of the hole with the charging sticks.

These charging sticks, made of hardwood and 5 ft in length, are fastened together with brass connectors of the patent knuckle-joint type made by the Twin City Iron Works, Hurley, Wisc. The connectors allow the charging sticks to be folded up or disconnected as they are withdrawn from the hole.

Each succeeding cartridge of dynamite is pushed into the hole until the required depth as determined by the loading chart is reached. It is usual to bring the dynamite charge to a point where it is about 3 ft from the two adjacent holes. Some of the holes, where they converge, are loaded nearer to the collar in order to cause proper fragmentation, when blasted.

The Primacord makes close contact with each cartridge of dynamite for the full length of each hole. When all of the holes are loaded, the Primacord lead from each hole is connected with a double half hitch to a short length of the Primacord, the end of which has a coupling for a No. 6 blasting cap attached to a 7 ft fuse.

In some holes, where the whipping action of the drill rods has enlarged them, the loading is difficult because the charging sticks slip by the dynamite cartridges and it is impossible to properly place the charge. Where these holes are encountered, waxed paper tubes, 5 ft in length and of the proper diameter for a snug cartridge fit, are solving the problem. The paper tube is filled with cartridges and the last one allowed to protrude about 4 in. so that the succeeding tube will fit over the end. In this way all the tubes in a hole are butted end to end for the length of the charge. The Primacord is punched through the first tube and fastened. As the tubes are pushed into the hole, the Primacord is in contact with each tube for later detonation. A final tamp on the last tube compresses the charge for close contact with the detonator cord.

In earlier blasting the dynamite used was E. I. du Pont de Nemours Company's 45 pct gelamite and the results were good. Later blasts, with 60 pct gelamite, seem to indicate better fragmentation, especially in ore where seams and slips are in evidence.

Performance Data

A careful record of all information was kept in the experimental work with this new method of blasthole drilling and stoping, for comparison with other methods and with other mines using the system. It must be said that the literature contributed by the many mining companies, both in the United States and in Canada, has been very complete in its details on performance and costs. We are glad to add our meager ex-

perience to that fund of knowledge for the use of the mining profession.

The following factors apply on blasthole diamond drill stoping on a tonnage of approximately 12,000 tons of ore.

Tons per foot of hole	4.21
Average footage per drill bit	85.49
Average feet per shift drilled	116 to 120
Tons per man (including driller, helper, scraperman)	42.20

It may be of interest to know that the record of bit footages shows 381, 185, 189, 158 and 144 ft for some of the pilot-type non-coring bits used. The pilot-type bit suffered severe salvage loss when running into rock walls or jasper seams.

The concave-type non-coring bit was the least successful of the three types used in the experimental work. Its use, over the period covered by this report, reduced the average footage per bit as shown in the tabulation above.

The coring bit, which seems to have found the most favor because it combines the feature of core recovery for ore-body delineation, and cheaper cost because of lower diamond weight, has shown footages of 166, 183 and 229 ft on some of the bits used. The salvage value on the used bits is higher than any other type.

Further trials at a later date with bits containing inserts of hard, alloy metals are to be made in an effort to further reduce the cost of the cutting medium which is the highest item in the total cost.

Costs

The matter of relative costs is of vital importance in the introduction of new methods in the winning of ores. It is only by actual experience in its use that we can arrive at an analysis of the worth of a new development in technique.

The records of our work covering this progress report show the costs as given in Table 1.

As shown in Table 1, the cost of diamond drill bits is the largest single item and constitutes 34 pct of the total cost per ton and 37 pct of the total cost per foot of drilling. This seems to check with the mass of literature on blasthole diamond drilling where a variation from 30 to 50 pct of the

TABLE 1—*Costs*

Item	Cost per Ton	Cost per Foot of Drilling
Labor:		
Drill operator	0.130	0.547
Drill helper	0.088	0.374
Scrapermen	0.042	
Materials:		
Diamond drill bits[a]	0.161	0.679
Explosives	0.036	0.151
Supplies, miscellaneous	0.018	0.075
Total	0.475	1.826

[a] The price of diamonds during this period was $8.50 per karat.

cost per foot is indicated. It is felt that future developments in bit materials will cause a reduction in the cost of the cutting medium.

Advantages

The diamond drill blasthole method was introduced at a time when it was difficult to obtain experienced mining labor. The Book mine was converted to underground mining methods during 1945 and the labor shortage which was caused by men being drawn into the military and naval services was still being felt at that time. It was thought that production could be maintained with the use of the diamond drill for blasthole drilling, in which method fewer men are used in the stoping operations.

The diamond drill method makes it possible to break large amounts of ore at a single blast and permits an even, regular production from stopes. It also makes it possible to prepare a number of drill rings in advance for future blasting so that production is not dependent on shift by shift breakage of ore.

The greatest advantage we have found in its use at the Book mine is in taking down rib pillars and back pillars after stopes have been mined out to their limits. I have in

mind one stope which is 100 ft long and approximately 40 ft wide in which a back pillar has been left over the stope to support a roadway in the wall of the open pit. This roadway is used for transporting materials by truck to the underground workings. A new roadway is to be provided away from the present mining operations. When this is completed, the back pillar over the aforementioned stope will be drilled out with the diamond drill and blasted down. This ore cannot be reached by ordinary methods without carrying out an extensive development in the rock walls of the stope to reach this back. The diamond drill can reach this ore with holes at a very low cost.

Recommendations and Summary

We can sum up our brief experience with this method of stoping in the following manner:

1. It has a useful application in stope blocks which require more than two subs in the standard sublevel method of stoping.
2. This method works best in firm, uniform stoping ground, free from water courses, seams and slip planes. Fractured ground results in blocky fragmentation which requires secondary blasting of ore chunks.
3. In drilling the blastholes, a high water pressure on the bit is necessary to flush out cuttings from long flat holes and reduce wear on the drill bits. With the coring bits used in soft iron ore, the high water pressure flushes out the core made in the core barrel and reduces blocking of the bit, so that pulling rods is infrequent.
4. Where it is possible to recover pillars left in the course of mining, the diamond drill has proved very efficient in their removal The experience of the iron ore operators in this type of ore recovery in the mines of northern New York is noteworthy, and can find similar application to the stope mines of the Lake Superior district.

Addenda

Following the presentation of the paper at the 20th Annual Meeting of the Minnesota Section of the AIME there were some questions from the members present. These are discussed in the following paragraphs.

Nature of the Ore

The iron ore at the Book mine is a semihard hematite with a rubbly or granular appearance in the broken state. Stope benches hold well and do not break away with an overhang of as much as 15 ft. Pillars between stopes retain the outlines established by mining for indefinite periods of time. The ore is not as hard as the high phosphorus ores in the Iron River, Michigan mines, but is harder than the subcaving ores of the Gogebic Range and the top-slicing ores of the eastern end of the Marquette Range.

Wandering of Drill Holes

There has been no noticeable variation in the direction of the diamond drill blastholes which have been drilled as deep as 60 ft. Occasional observations have been made of the imprint of holes left after blasting in the back of the undercut in the stope block and no appreciable variation in direction is noted.

Other Drilling than Horizontal Plan

The paper presented is a progress report which covered a program of drilling using the horizontal plan for blasthole purposes. Inclined holes have been tried out and a new stope is now being developed for vertical ring drilling. It is evident from the amount of work done thus far that the diamond drilling method is very flexible and may be adapted to almost any type of drilling plan, whether it be horizontal, vertical, up or down or at any angle desired.

Tests on Hard Alloy Insert Bits

Mention was made in the paper that tests were being conducted with bits con-

taining inserts of hard alloy metals for the cutting medium.

The test reported to date on the use of the special alloy bits shows a drill bit cost per foot of \$0.082 and it appears that even lower costs per foot for the drill bits will be indicated after they have been dressed and re-used.

It will be noted that the drill bit cost has already been reduced from \$0.679, as quoted in the paper, to \$0.082 per foot by the use of Kennametal tungsten carbide insert bits in ore drilling and that lower costs are indicated which will reduce the total cost per foot to a little over a dollar per foot. Reports obtained on the use of Kennametal bits, subsequent to the presentation of this paper, resulted in costs of \$0.040 per foot for drill bit expense. The drill operators report that they have less trouble with the blocking of the bit when using the hard alloy insert bits. The serrated edge of the cutting face apparently breaks up the small amount of core which is made in drilling and the high water pressure flushes out the cuttings.

Anaconda's Operation at Darwin Mines, Inyo County, California

By Dudley L. Davis* and E. C. Peterson,* Members AIME

(Los Angeles Meeting, October 1947)

Introduction

The Darwin District is 30 miles east of Olancha which is 220 miles north from Los Angeles via U. S. Highway No. 6. The ore deposits occur in the Darwin hills that have been elevated above the Darwin Plateau by typical basin and range-type faults. Some of these faults have been active quite recently as shown by the displacement of Quaternary (?) basalt flows and the rejuvenation of erosion in Darwin wash. The ore bodies occur as fissure fillings and replacement of favorable beddings in a thick series of Paleozoic limestones, dolomites, shales and quartzites.

Rich oxidized lead-silver ores were discovered in the early seventies. By 1875, Darwin had a population of about 5000, and by 1880 several small mills and smelters had been built. Early exhaustion of surface ores, the isolated location of the district, and fluctuation of metal prices caused intermittent operations until World War II. The district production is estimated at about $3,000,000 prior to 1900 and perhaps $4,000,000 between 1900 and 1945 when Anaconda purchased the major producing mines.

For the past two years, daily production has averaged 75 tons of direct shipping and 150 tons of milling-grade ore. At present, 300 tons of mixed oxide and sulphide ore are treated by flotation, the oxidized lead minerals being activated by addition of sodium sulphide after galena and sphalerite have been recovered from the circuit. Extraction averages 85 pct of the lead, 80 pct of the silver, and about 33 pct of the zinc in a bulk concentrate. The mill is designed so that minor changes permit selective flotation of pyritic lead-zinc ores encountered at depth.

General Geology

The oldest rocks in the Darwin hills are a series of Paleozoic limestones, dolomites, shales and quartzites probably Pennsylvanian in age. These rocks have been intruded on the west by the Coso granite batholith and on the east by a granodiorite stock. Numerous sills and dikes grading from orthoclase granite to gabbro are exposed in the underground workings.

The sedimentary rocks have been considerably folded and faulted, the most prominent structure being a northwest pitching anticlinal fold, the crest of which lies just west of the ridge line of the Darwin hills. The east limb of this fold has been intruded by the granodiorite stock and east of the stock the sediments show several closely spaced anticlines and synclines. The minor flexures on the west flank of the major fold are structurally important in localizing the ore in the Defiance and Essex mines of the Darwin group.

There are three principal systems of faulting and fissuring, most of which are post-intrusive and premineral in age, with minor amounts of postmineral movement. The Darwin tear fault is the largest, most persistent fault in the district. It strikes N 60 to 75°W, dips steeply south and is traceable for about 10 miles. It is a normal

Manuscript received at the office of the Institute Nov. 24, 1947. Issued as TP 2407 in Mining Technology, July 1948.

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fault with a horizontal shift such that the north side of the fault moved west as well as upwards relative to the south side. Its relationship to the intrusive rocks is not clear but smaller parallel faults of similar nature cut the intrusive and so it is assumed that the larger fault is also post-intrusive.

A second system of fissures strike N 50 to 70°E, show very little displacement and are commonly mineralized. A third set strikes N 10 to 40°E and are both premineral and postmineral in age.

The mineralogy of the deposits is most interesting from a collectors standpoint, as there are many rare minerals and mineral associations present. Genetically, the ore may be divided into three general classes: replacement of silicated limestone (tactite) rocks, bedded replacements, and fissure fillings.

There are four types of ore to be mined, high-grade oxidized silver-lead ores, milling-grade oxidized lead-silver ore, sulphide lead-zinc milling ore, and high-grade tungsten ore.

Mineralogy

With a few exceptions of massive veins and bunches of galena, the high-grade ores are strongly oxidized. Enrichment by subtraction of zinc, sulphur, and some iron during oxidation is most important. Sometimes silver has been transported to be reprecipitated as sooty argentite and cerargerite. Cerussite, anglesite, plumbojarosite and galena are the principal lead minerals found in the oxidized areas; accessory minerals are iron oxides, jasper, clays, sulphur, gypsum, jarosite, calcite, hydromica, quartz, and fluorite.

Some of the rarer minerals found in the zone of oxidation are wulfenite, vanadinite, linarite, caledonite, hydrozincite, pyromorphite, and brochantite, together with azurite, hemimorphite, and malachite.

One end of a high-grade oxidized lead-silver stope was found to assay 10 to 15 pct WO_3. Several hundred tons of this type ore have been mined and many well-formed crystals of scheelite have been taken by collectors. These crystals are found lying loosely embedded in a matrix of iron oxide, jarosite and clay. There has been considerable discussion regarding the origin of these crystals but the author is of the opinion that they were deposited at an early stage of mineralization related to the intrusive granodiorite and that later lead-silver mineralization took place along the same zone of weakness and engulfed the sheelite crystals.

In the ore zones where oxidation has not taken place, ore minerals are sphalerite and galena with minor amounts of chalcopyrite and tetrahedrite in a pyrite, fluorite and calcite gangue. The limestone is usually completely silicated to a mass of garnet, wollastonite, diopside, epidote, and quartz.

Mining

Mining methods are not standardized but are adapted to suit conditions existing in each ore zone.

Square-set stoping is employed in the loose, heavy, high-grade areas where only a small amount of ground can be opened up without timber and selective mining is necessary. In large replacement ore bodies, pillars are left between the square-set stopes, and the pillars later are recovered by a modified Mitchell top slice. A modification of the slot method of stoping, as developed by the Anaconda Copper Mining Co. in Butte, Mont., is used in the square-set stopes, with the main difference being that waste fill is seldom used.

Where the walls are hard or the vein quite narrow, open or rill-type stopes are common and large areas of sulphide mineralization having strong walls are mined by shrinkage or sub-level methods.

The ore bodies occur in very hard tactite areas so that waste development work is done in the hardest of rocks. Practically all development and mining are done on a

contract basis. Drilling in drifts and crosscuts on the main haulage level is being done with Sullivan hydro-drill jib with automatic DA-35 drifters. Electric haulage, and modern mechanical equipment are used throughout the mine.

For a 230-ft vertical development raise, a diamond drill hole was drilled to serve as a pilot for the raise and to furnish ventilation. The time saved in supervision and the speed with which the raise was run more than paid for the cost of the drill hole.

The ground is so hard that shaft sinking, both vertical and inclined, must be done with automatic drifters on a crossbar.

Details of the principal mining methods used at Darwin can be most easily explained by presenting the history of what we call the 430 ore body. The part that has been mined was shaped like a truncated discus on edge, with the haulage level acting as the truncating plane. The discus was roughly 150 ft long at the haulage level, reached a vertical height of 150 ft and dipped approximately 55°. The unmined part of the discus extends below the level, where it is being developed for future production.

When the shape of the ore body was determined at the level, it was found to be roughly crescent shaped, with a thickness of 40 ft at midsection and tapering to zero in thickness at 75 ft in each direction. It consisted of a gossan-like mass of thoroughly oxidized material resembling ashes, containing bunches of lead sulphate, carbonate and residual galena. Level samples ran from 1 pct Pb, and 1 oz Ag in the material predominantly iron oxide, to 35 pct Pb and 20 oz Ag in the better areas.

The thickness of the ore body at midsection and the fact that it projected into an unexplored area presented a problem in determining a method for mining. Such ore bodies are generally encased in a shell of extremely hard silicated lime, but occasional pillars are needed to keep waste from scaling from the hanging wall when fairly large areas are open. Also, the ore appeared spotty, therefore, a system had to be chosen that would provide segregation of mill ore from shipping ore. It was decided that a slot stope of 3 square sets along the strike and reaching from footwall to hanging wall (which amounted to 8 square sets) would develop the ore body and give a cross-section of values. In the meantime, a crosscut was started to intersect the slot stope 150 ft vertically above the level. The slot stope revealed shipping-grade ore to the top of the 5th floor, poor mill ore for the next 5 floors, a mixture of mill ore and shipping ore to the 16th floor, then 5 floors of massive anglesite and galena to the top of the 20th floor where the bedding rolled over and terminated the ore. The ore remained 8 sets thick to the 13th floor, then gradually thinned as the footwall steepened and met the hanging wall at the top of the 20th floor.

As the shape of the ore body was gradually revealed, a second and third slot stope were started, each leaving a pillar 7 square sets long toward the central stope, with fringe areas in opposite directions. All three stopes were oriented along the strike in a parallel position so that stopes could eventually connect without misalignment of sets. Connecting sublevel drifts were driven where needed (see Fig 1).

The "hood" of massive galena and anglesite had to be mined from the top down because its excessive weight would "wreck" the stope if shot into the square sets. (Filling for square sets could not be economically obtained.) Therefore, a connecting drift was driven along the 16th floor and the heavy ore from above was broken to the floor of the drift where it was reduced to approximately 8-in. material, scraped over to chutes in the slot stope and produced to the level. Sills of 10-ft were carried on the 16th floor and the stope was caught up tightly with square sets, as all the ore above this floor was mined. It

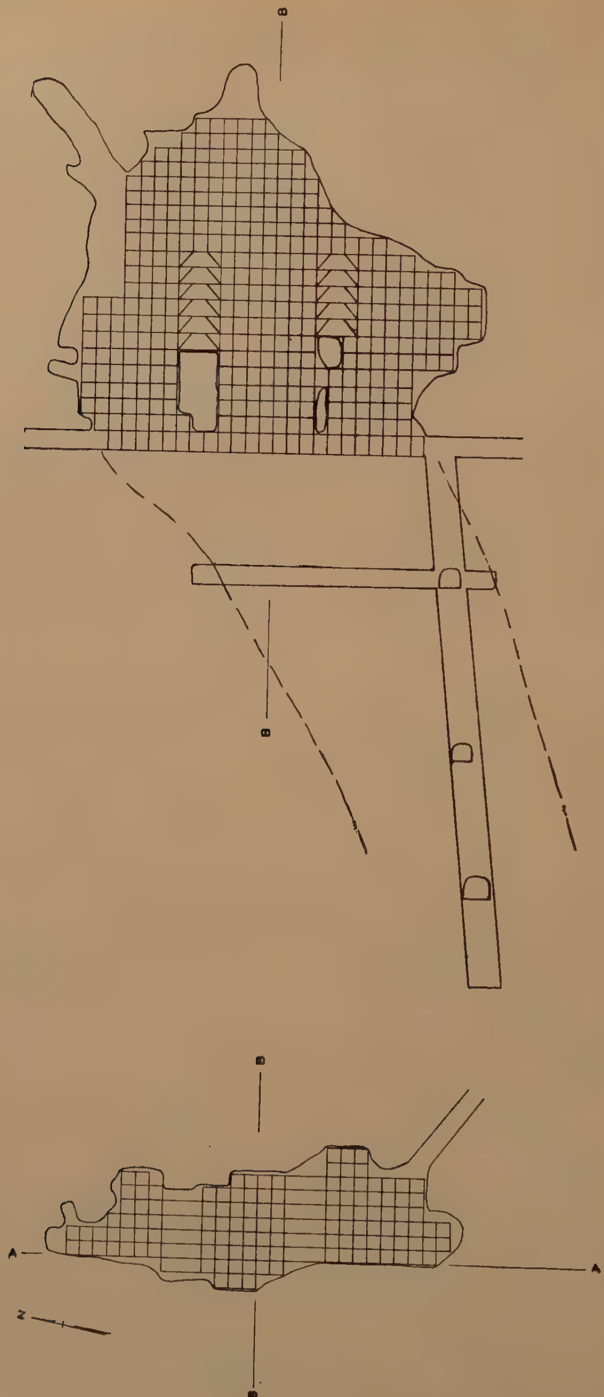


FIG 1—THE 430 ORE

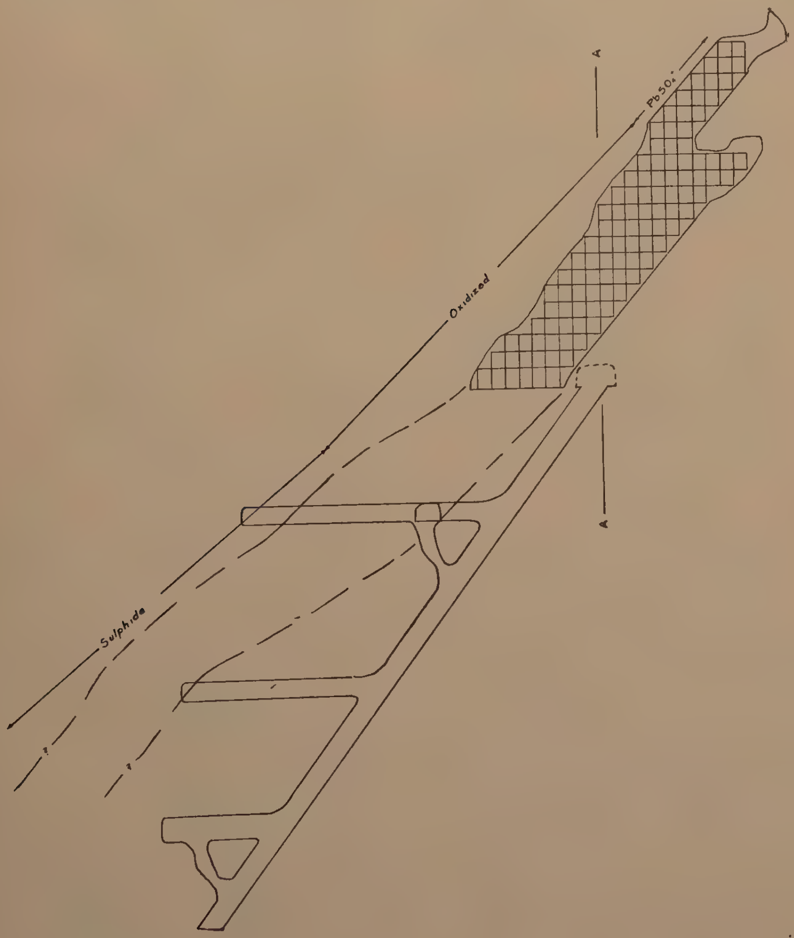
amounted to a slice 5 square sets high across the top of the stope.

A second slice was started by driving connecting square sets along the hanging wall from one slot stope to another on the 14th floor. Again 10-ft sills were carried and the slice, 2 square sets thick, was silled off with regular square sets that caught the 10-ft sills of the 16th floor. A third slice, which included the 12th and 13th floors, was mined in the same manner and completed the extraction of heavy lead ore.

During the period that heavy ore was being mined from the top of the ore body, fringe areas bordering the two outside slot stopes were mined as open shrink and stulled areas or by means of square sets, according to requirements.

The ore body at this point was stripped of its fringe ore and its high-grade "hood"; all that remained was the ore in two pillars 7 sets long by 8 sets from footwall to hanging wall, extending from the main level to the 12th floor. Thin slabs were loose at points along the hanging wall, but the large network of unfilled square sets was standing remarkably well.

The next step was to slice 8 vertical rows of square sets from the pillar faces, cutting the pillars to a thickness of 3 square sets.



BODY, DARWIN MINES.

This was accomplished by extending each set in the row from the level to the hanging wall or to the top of the pillar; starting with the hanging-wall set and retreating toward the footwall. No chutes were carried above the first floor and broken ore was dropped through open sets to a chute-mouth on the level.

After two slices had been taken from each side of both pillars, the timbers and stope walls continued to hold, therefore, it was decided that it would be safer to Mitchell the pillars from the 16th floor to the level rather than try to mine them from the level up. This was successfully accomplished, and all of the remaining ore was recovered except minor tonnages of low-grade mill ore, which were left for economic reasons.

Total extraction of the ore body amounted to approximately 40,000 tons, half of which was segregated as shipping ore, the rest going to the mill.

General

Slot mining as utilized at Darwin is patterned after the method developed at Butte, Mont., and described by L. F. Bishop.[1] The differences are that at Darwin, square, 8 by 8 Oregon fir timber is used, and most stopes are not filled with waste.

A second large ore body on the Essex zone was mined by extracting panels from either side of a central raise. Mining progressed in a downward fashion from the surface a distance of approximately 500 ft. Panel sizes were roughly 50 ft along the strike and 50 ft down the dip and 4 square sets from footwall to hanging wall. They were mined by slot-method square sets, or open-stull mining to the sublevel below. Ore was then slushed to the central raise for production. Fill or caved stope mats were kept overhead and were caught from below as mining progressed (see Fig 2).

Open square-set mining has proved successful at Darwin but cannot be recommended where ground takes weight beyond superficial slabbing. The spotty nature of Darwin ores makes it necessary to timber in order to segregate valuable high-grade ore from mill ore or waste. It costs nearly $20 to ship and smelt a ton of ore, therefore, it is important to keep waste out of the direct shipping ore.

Milling

At Darwin mines, 40 miles southeast of Lone Pine in Inyo County, Calif., Anaconda Copper Mining Co. is operating a mill of 150-tons daily capacity for the treatment, at present, of oxidized lead-silver and partly oxidized lead-silver-zinc ores that occur in tactites or silicified limestones as replacement deposits.

This mill was built in 1941–1942 and was used by the former operators for the selective lead-zinc oxide-scavenger flotation treatment of low-grade, slightly oxidized lead-zinc ores. Soon after being taken over by Anaconda in August 1945, the milling scheme and flowsheet were changed to treat oxidized and mixed oxide-sulphide ores by sulphidized flotation.

In consequence of the lower ratio of concentration obtained when treating such oxidized ores of higher grade and because of the slimy nature of the concentrate, the incapacity of the former filter plant was immediately apparent. It was decided to build a new filter plant of sufficient capacity and to provide for mechanical handling of the filtered concentrate to bins for direct loading into 25-ton semitrailer dump-trucks. Provisions were included for the testing and erection of a rotary drier, if such became necessary to lower the moisture content of the product before shipment. However, filtration alone has proved adequate.

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[1] L. F. Bishop: Vertical Slice and Slot Stoping at Butte. *Trans.* AIME (1945) **163**, 262.

Fig 2—The Essex ore zone, Darwin mines.

It may be of interest to mention here that as an experiment while the filter plant was being erected, and taking advantage of the extremely dry climate of the region, the concentrate was dried in 8 solar ponds 20 by 100 ft and 20 by 200 ft in areas that were excavated in a caliche side-hill slope adjacent to the mill. Concentrate pulp at approximately 50 pct solids was pumped into the solar ponds to a depth of about 3 ft. Individual large ponds held as much as 575 dry tons of concentrate. In summer the moisture content was reduced to about 11 pct in 40 to 50 days but in winter the drying period to obtain the same decrease in moisture content extended to 90 to 100 days. However, by this method a large inventory of concentrates was carried and when concentrates were reclaimed using a power shovel and hand cleanup much dilution from the bottoms and side banks was experienced.

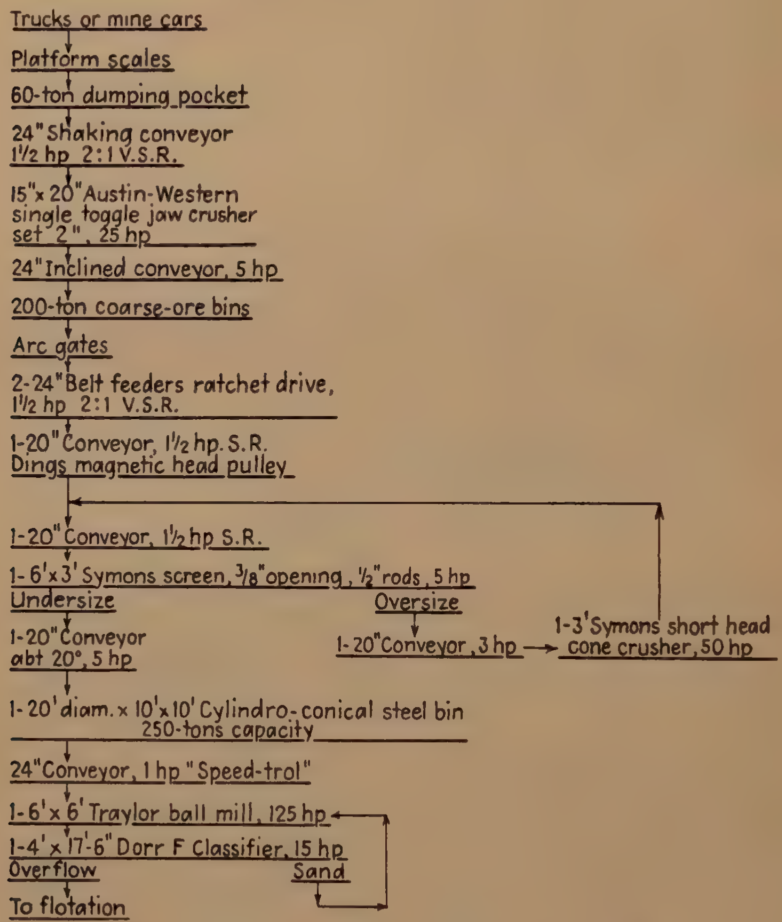

FIG 3—CRUSHING AND GRINDING FLOWSHEET, DARWIN MINES.

With the filter additions, erection of automatic samplers, and other improvements, the flowsheet is given in Fig 3 and 4. Fig 3 shows the crushing and grinding sections and Fig 4 shows the flotation and dewatering sections of the mill flowsheet.

Size reduction in crushing is to −2 in. in the jaw crusher and to −38 in. in the Symons crusher operating in closed circuit with the Symons screen. Flotation feed has

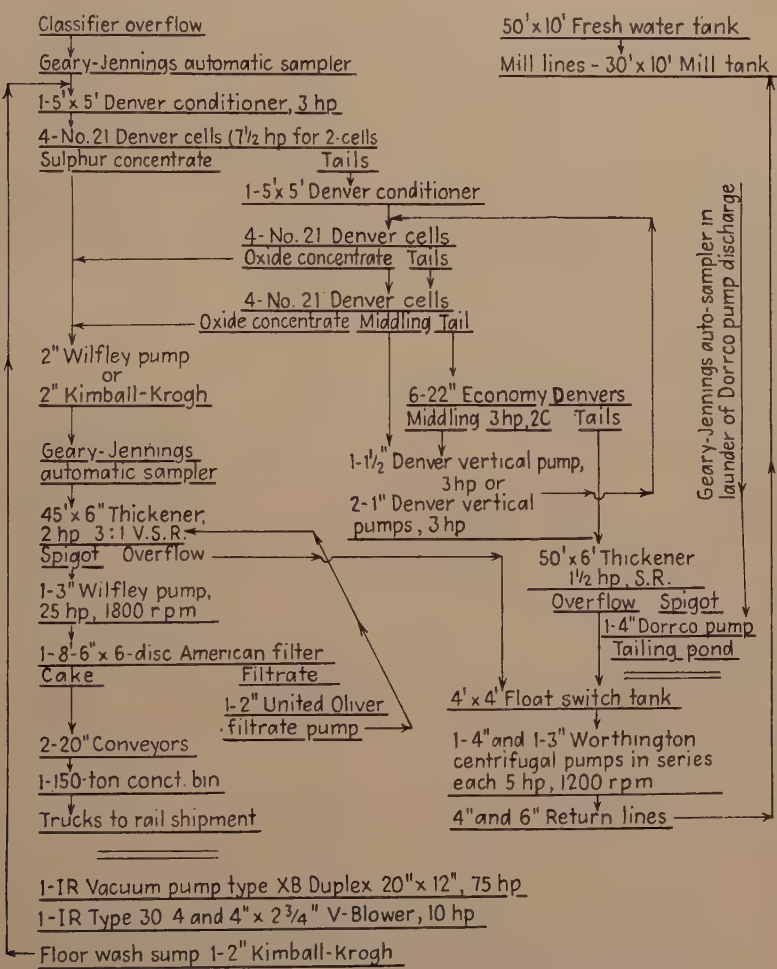

FIG 4—FLOTATION AND DEWATERING FLOWSHEET, DARWIN MINES.

approximately the screen analysis shown in Table 1.

The nature of the ores received as mill-feed is widely variable from the soft gougy iron-oxide jarosite ores, through the usual oxide-lead ores, to the extremely hard and dense flint-like tactite disseminated ores. Crushing and grinding of the tactite is sometimes difficult, and the gougy ores, when damp, give trouble in fine crushing. The proportion of 4 to 2 in. balls charged daily to the ball mill is varied by periods to suit the changing hardness of the mill feed.

The principal sulphide and oxide-ore minerals present in the ore are: cerussite, galena, anglesite, argentite, plumbo-jarosite, sphalerite, marmatite, smithsonite, and hydrozincite. Many minerals of minor occurrence are found or observed in the ore: chalcopyrite, tetrahedrite, linarite, caledo-

nite, mimitite, pyromorphite, vanadinite, and wulfenite. Garnet, wollastonite, jarosite, quartz, fluorite, pyrite, limonite, gypsum, and clays are some of the gangue minerals.

TABLE 1—*Average Screen Analysis of Flotation Feed*

Screen Size	Per Cent	Cumulative Per Cent
Plus 48 mesh	0.42	0.42
Plus 65 mesh	1.12	1.54
Plus 100 mesh	4.09	5.63
Plus 150 mesh	6.97	12.60
Plus 200 mesh	12.21	24.81
Minus 200 mesh	75.19	75.19
	100.00	

Successful flotation treatment of oxidized and partly oxidized lead ores requires the removal of the sulphide minerals before sulphidization in consequence of the depressing effect of sodium sulphide upon the flotation of sulphide lead and silver minerals. Therefore, by the addition of xanthates, sodium silicate, and frother, the readily floatable sulphides, including any sphalerite contained in the ore, are removed. Subsequently, the oxide-lead minerals are sulphidized and floated by stage additions, from cell to cell, of sodium sulphide, sodium silicate, and pentasol amyl xanthate. Additions of frother are rarely required.

Average reagent consumptions over the period of a year and the place of addition are given in Table 2.

TABLE 2—*Average Reagent Consumption*

Reagent	Lb per Ton Ore	Place of Addition
Sodium cyanide	0.32	Ball mill.
Z-3 xanthate	0.57	Conditioner No. 1, Oxide cell No. 1.
Z-6 xanthate	2.67	Oxide cells Nos. 1-2-3-4-6-8.
Sodium sulphide	10.35	Oxide cells Nos. 1-2-3-4-6-8.
Pentasol No. 26	0.22	Conditioners Nos. 1 and 2.
Lime, hydrated	0.33–1.00	Concentrate and tailing thickeners.
Steel balls, 4-in.	0.20–1.10	
Steel balls, 2-in.	0.90–1.85	

Sodium cyanide added to the ball mill has two functions; to depress barren pyrite into the tailing, and/or to decrease the quantity of sodium sulphide required for the sulphidization of certain ores. In the oxide circuit the bulk of the reagents ordinarily is added to the first few cells, with the quantity staged tapering off to the end of the circuit. The additions of reagents to the last cells of the circuit are increased in quantity for the treatment of refractory anglesite ores. Sodium silicate functions to disperse the slimes and to overcome the over-flocculation caused by the addition of large quantities of sodium sulphide.

Over a twelve-month period the results shown in Table 3 were obtained. More recently, similar results have been maintained in the treatment of lower grade development and dump ores.

TABLE 3—*Assay and Distribution*

	Weight, Pct	Assay				
		Pb, Pct	Ag, Oz.	Zn, Pct	Fe, Pct	Insol. Pct
Heads		10.39	6.97	4.53		
Concentrate	24.14	36.61	23.54	6.03	10.4	13.0
Tailing	75.86	2.05	1.87	3.98		

	Distribution		
	Pb, Pct	Ag, Pct	Zn, Pct
Heads	100.0	100.0	100.0
Concentrate	85.04	80.03	32.54
Tailings	14 96	19.97	67.46

Thickened tailings at approximately 55 pct solids are elevated through the spigot of the thickener by a 4-in. Dorrco diaphragm pump to the tailing launder in which the tailings flow by gravity to disposal behind earth dams just above the bed of Darwin Wash.

Water for milling and all camp purposes is obtained from shallow wells in Darwin Wash about three miles from camp. At

present this water is being lifted 1800-ft vertically in two nearly equal stages to 3 50,000 gal storage tanks above the camp and mill and from which the mill fresh-water tank is supplied. Approximately 30 gpm of fresh water is demanded by the mill. Water reclaimed from the concentrate and tailing thickener overflows constitutes more than 50 pct of the total mill-water demand.

Expansion Program

Present construction, which is expected to be complete and in operation before the end of the year, is in progress to expand the nominal capacity of the mill to 300 tons per day and to provide a flexible flowsheet for alternate operation for the sulphidized flotation treatment of oxide-lead ores or the selective flotation of lead-zinc sulphide ores being developed at depth in the mine.

This expansion includes the erection of a 300-ton concrete and steel crude-ore bin, a 36 in. by 25 ft pan feeder, a 24 by 36 in. jaw crusher, and changing to a coarse bowl in the Symons cone crusher. By this arrangement secondary breaking in the mine and on bin grizzlies will be unnecessary or held to a minimum.

An 86 Marcy ball mill and an 8 by 23 ft-4 in. Model D Dorr duplex classifier now in place will grind the whole tonnage. The present grinding equipment will be held as standby or for further possible expansion. Additional equipment will increase the flotation capacity. Another 8 ft-6 in. by 6-disc American filter will be erected over a new steel concentrate bin. Disposal of oxide tailing and reclaiming of water will require the erection of a new 60-ft thickener with all the necessary auxiliary equipment. This thickener will also be used alternately, as required, for lead concentrate from selective lead-zinc flotation.

Additional water has been developed in Darwin Wash and the pumping plant and system of piping will be improved to double the present pumping capacity and to make the 1800-ft vertical lift in one stage.

Acknowledgments

The writers wish to express appreciation to Mr. V. C. Kelley whose paper on the Darwin District has been used in the preparation of this report, to Prof. Paul F. Kerr for his work on the tungsten mineralization and determination of the clay minerals, and to the Management of the Anaconda Copper Mining Company for permission to publish this paper.

Drilling and Blasting Practices Past and Present at Bingham Canyon Utah Mine, Utah Copper Division of Kennecott Copper Corporation

By Richard H. Willey*

(New York Meeting, March 1947)

Efficient handling of large tonnages in open-pit mining demands, primarily, an abundance of well fragmented rock. To accomplish this, a drilling and blasting department composed of a well-knit, versatile group of workers and supervisors must be established. Some costs will remain relatively on the same level over a period of years, while others can be greatly lowered as will be shown. The Utah Copper Division, Kennecott Copper Corp., Bingham Canyon, Utah, has been ever alert for improvements in operating methods and equipment to gain increased efficiency and lower costs.

Bingham Canyon Mine

The greatest dimension of the mine is 8000 ft northeast-southwest with a width of 5000 ft southeast-northwest. From the lowest level, at an elevation of 6040 ft above sea-level, the mountain rises in 27 benches on the west side to W level (elevation 7800 ft) and 21 benches on the east side to Q level (elevation 7460 ft). At the inception of open-pit mining in 1906, the overcapping of leached porphyry had to be removed to reach a commercial grade of sulphide ore. The overcapping or waste now remaining is quartzite with intercalated limestone beds. The limestones are usually soft in nature except in the south-middle portion of the mine where they have been metamorphized along the contact zone. This area usually gives drilling and blasting operations a great deal of trouble. Here occurs the greatest amount of secondary blasting, as the well-established fracture planes cause the altered beds to break into large boulders. Several massive quartzite "ribs" occur in the overburden, and as a result, the uniform breakage is difficult to obtain. The major quartzite areas have been subjected to complex folding, fracturing and faulting because of the intrusion of the underlying stock. The monzonite porphyry, which carries the commercial values, presents no serious drilling problem because of its softness and extensive minute fracturing throughout the mass. A series of major slips or faults pass through the ore body which, although of considerable advantage to the drilling and blasting department, sometimes works to the disadvantage of the track department. Drilling and blasting of the porphyry apparently becomes easier as depth is attained, but highly siliceous streaks throughout the pit still present difficulties. At no place is the rock of such character as to allow excavation without preparations. On certain higher elevations or levels, old fill material can be dug by the shovels, but only at some risk to equipment because of bank slippage.

Past Methods

At the outset of shovel operations in 1906, drilling was done with the Ingersoll-Rand F-24 piston drill mounted on an un-

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Manuscript received at the office of the Institute March 14, 1947. Issued as TP 2271 in Mining Technology, November 1947.

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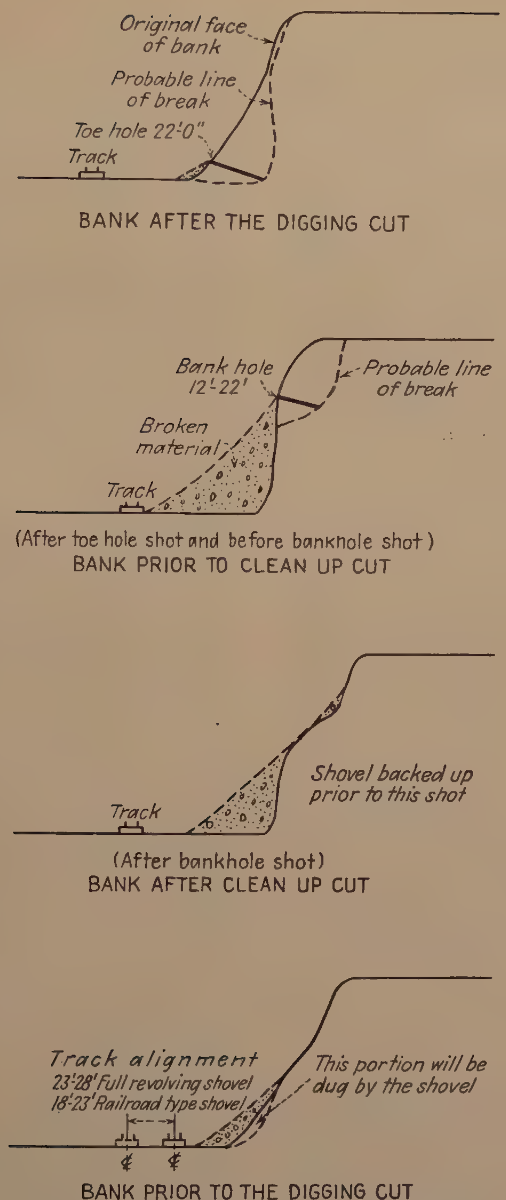

Fig 1—Old methods of fuse blasting.

weighted tripod which rested on three ordinary railroad ties. The drilling grade was dug by hand and loose muck was removed to present a solid drilling face. The machine was a cumbersome piece of equipment with a total weight of 540 lb for the drill and tripod. The drill steel was solid (the water was poured in the hole by hand) and came in 2-ft changes with the starter steel being 3 ft and the finishing steel 23 ft long. This required the use of 11 pieces of steel for the completion of one hole. The drill hole was inclined from 5° to 15°, or enough to bottom the holes at grade. Initial bit gauge was 3½ in. with a ⅛-in. gauge reduction for each change; thus, the finishing drill-steel gauge was 2⅛ in. Holes were spaced from 15 to 25 ft depending upon the ground and bank condition.

As the shovel moved across the level on a digging cut, a drilling crew, or crews, would closely follow the shovel travel and drill the necessary toe holes (Fig 1). Springing and loading crews, coming behind the drilling crew, would blast the bank as required. Seldom were the holes left loaded for more than three days. This offered a blasted bank or clean-up cut to the shovel any time it became necessary to turn around or back up, which usually was at the end of the level. When the shovel turned around or backed up, drilling crews immediately ascended the muck pile to drill bank holes in front of the shovel, which were blasted into the pit as the shovel advanced.

Springing and Loading

It is in springing and loading operations where men of long experience are required. The ability to recognize ground, to know how to spring it, the quantity of explosives it will require, and to estimate the size of the chamber, is indeed a fine art. Some ground will chamber readily with 3 springs of 5 sticks, 20 sticks and 50 sticks, respectively, which provides room for 250 to 300 lb of powder, while other ground may need 10 springs, with the final spring using 100 sticks to prepare for the same amount of powder. The original procedure utilized approximately 5 springs with 60 sticks on the final spring. All the powder for springing and loading was inserted into the hole through an iron pipe 1½ in. id and 23 ft in length. The cartridges were forced through the pipe with a single-piece tamping stick 25 ft in length. In springing, only water was used for stemming, the ratio was approximately 1 gal per 10 sticks of powder. This was poured down the pipe after the springing cartridges were in the chamber. Following the water, the primer cartridge with a 2½-ft fuse was "spit," then inserted into the chamber. The loading pipe was then withdrawn.

Permission to "spit" was obtained from a centrally located watchman by blowing a whistle at the place of springing. He gave permission by sounding a general warning whistle of his own. Red flags on the edge of the bank indicated to workmen in the vicinity the location of springing operations. Hercules No. 4, a 65 pct ammonia-nitrate explosive by weight, was used in all dry holes, both for springing and main charges. On levels in which water was encountered, Hercules Extra L. F., a 40 pct ammonia-nitrate explosive by weight, was used. Detonation of the charges was accomplished by inserting double-primed cartridges with 6-ft fuses into the holes. Three crews usually worked in tandem starting from one end of the blast and working to the other end. If any hole had a tendency to "throw," it usually was prevented by the blanketing action of the preceding blasted hole. Fragmentation usually extended ½ to ⅔ of the distance up the bank depending, of course, upon the height of the bank. On especially high banks, sometimes two tiers of bank holes had to be blasted. Careful control of the blasted material was constantly kept in mind. The short reach of the old railroad-type shovel did not afford a great deal of pit room. Tracks

which were covered had to be cleaned by plows or by hand.

Present Day Methods

From time to time as a result of continual experimental efforts, new methods were proved and incorporated into the drilling and blasting operations. These new methods in conjunction with larger shovels and the use of bulldozers now afford heavier blasting and greater track alignment. At present, a drilling crew consists of two or three men. In average ground, the use of a third man on the drilling crew to dig grades, obtain material and give general assistance to the driller and helper, increases the work accomplished proportionally. Where only one shift is worked by the drilling and blasting department, in contrast to two shifts for some shovels, it becomes necessary to use three-man drilling crews.

Compressed air is supplied from the three compressors, each of which has a rated capacity of 3720 cu ft of air per minute. Main feeder lines (varying in size from 12 in. at the compressor to 4 in. at the extremities) completely encircle the pit, from which 2-in. lines extend across each level to join the feeder line on the other end. Numerous large air receivers are distributed along the main line at strategic points. Air hoses (1-in.) are passed under the track from the level lines to supply the drilling machines, which now are the Ingersoll-Rand DA-35, an automatic power-feed machine with the sliding cone. The old tripods were converted to use with the new machines; the three ties, upon which the tripod rests, are now half-thickness railroad ties. By using the sliding cone, an easy 4-ft change is afforded, the starter steel being 4 ft and the finishing steel 24 ft; 6 pieces of steel comprising the set. The steel is hollow for use with water, and detachable bits are of the Ingersoll-Rand 3-in. Type 2. It is a matter for conjecture whether these bits prove more satisfactory than the Type 2, Ingersoll-Rand bit, some of which are still in use. From outward appearances, the Eagle Picher, or A-2 Type, seems the best, especially in graveling ground where steel removal causes difficulties in drilling speed, and in gauge maintenance. These bits are reground to gauge and sharpness in the company shops. An average of 3.2 regrinds are obtained per new bit.

Cold weather works hardships on drilling operations. The barrels of water, which are filled by a special water car each night, usually are frozen at the start of the shift. Pressure tanks are warmed by fire, the air and water must be heated by putting 1-in. air and water pipes on the fire, and machines have to be thawed out to give full striking power. Delays on extremely cold mornings sometime consume as much as 2 hr. Water lines have become frozen solid during the 10 or 15 min required to move the outfit from one grade to another.

The spacing of drill holes is determined by the type of ground and condition of the bank. Usually the foreman marks the location of each hole. Holes are spaced from 20 to 25 ft and are kept as close to grade as possible, with sufficient pitch to bottom the hole at grade. If there has been inadequate shovel clean-up or sloughing of the bank, the holes have to be moved up the bank. This greatly increases the possibility of a hard toe or high point on the level grade. In some ground, relief holes must be drilled on the shovel grade to rectify this abnormal condition.

Wagon Drilling

Gradually at first, but gaining momentum, especially in the last two years, the drilling of bank holes has decreased to be replaced by vertical wagon-drill holes. The large increase in cubic yardage per manshift can be greatly attributed to this operation. A comparison between banks blasted by bank holes and downholes shows greater yardage per manshift in favor of the downholes. It is only where banks are of such

height (and there are some over 70 ft) that downholes cannot reach the objectionable bulge or overhang. Under these conditions, bank holes are a necessity. The average bank can be given a good slope by combined toe holes and 24-ft downholes on top. A flatly sloped bank affords greater safety to the trimming operations and reduces the accident hazard to workmen and equipment. Downhole blasting has the effect of smashing down the crest or edge of the bank; whereas a bank hole has a tendency to force the face of the bank out, leaving a treacherous shattered mass which, at times, hangs up on an underlying solid rib.

The wagon drill in use is the Ingersoll-Rand X-71, mounted on a three rubber-tired wagon drill carriage. The steel and bits are interchangeable with those used on the DA-35. Grades for the wagon drill are dug by bulldozers, solid rock usually is reached to permit easy collaring of the hole.

Departmental Cooperation

With the installation of full-revolving shovels in recent years, the problem of selective mining has been greatly facilitated. Shovels are turned around, moved ahead or backed up to excavate high-grade or low-grade ore, or overburden, depending upon what is required. Consequently, drilling and blasting schedules must be sufficiently flexible to accomodate changes in shovel operations. Close cooperation between shovel and blasting departments makes these coordinated workings feasible and easily arranged. Bulldozers from the track department aid in establishing drilling grades at the toe and in uncovering buried holes. Loose material which hinders wagon drilling is also removed by bulldozers.

Distribution of Material

All sharp bits, new or repaired drill steel, air hoses and other required equipment are delivered by railway flat car or motor car. Each day telephone orders are received at the drill shop for materials or equipment needed for continuation of drilling and blasting operations. These are ordered by each level drill foreman for his respective territory which usually comprises three or four levels. Materials are delivered each morning to the east side and lower pit levels, and to the west side levels each afternoon. Worn or damaged equipment is picked up at delivery points, to be returned to the drill repair shop.

Explosives

Two types of Hercules explosives are presently in use. In late years and prior to 1946, Gelamite No. 1 was used exclusively for blasting, and proved to be a well-chosen, versatile powder. It is a 65 pct weight strength semi-gelatin powder with a detonating speed of 13,500 ft per second and is delivered in the conventional 1¼ by 8-in. cartridge. Springing, blockholing, "dobying," trimming and a major portion of the blasting is accomplished with this powder. Its resistance to water is high.

Within the last six months, Hercomite Bag, a Hercules Powder Co. product, has been put in use. This is a free-flowing high-ammonia powder delivered in 50 lb boxes containing four 12½-lb bags. The weight strength of this powder is 65 pct with a detonating speed of 5000 ft per second, a powerful but slow explosive. All vertical holes, with the exception of wet holes, are loaded with Hercomite Bag, as the ammonia powder is adversely affected by the presence of water. Prior to the experimental use of Hercomite Bag, a fast action powder was deemed best; observation of blasting areas indicates that a slow acting powder is giving better than average results. Toe holes shot with Hercomite Bag result in a surging or heaving action into the mass surrounding the charge instead of the quick-snapping action or wedge shaped pulling of material at the base of the bank as exhibited by the use of Gelamite. Overbreak or backbreak is increased sometimes

as high as 50 pct with the likelihood of holes "throwing" being greatly decreased. Fragmentation apparently is just as good if not better, even in massive, blocky ground. Vertical wagon-drill holes exhibit even better results with the Hercomite Bag. Springing is lessened, loading is faster by 100 pct and sloping of the bank is better by the increased backbreak. Since the Hercomite Bag or free-flowing powder has been in use only a short time, accurate results have not, as yet, been definitely determined.

Storage and Transportation of Explosives

Close proximity of the mine to Bacchus, where the Hercules manufacturing plant is, precludes the necessity for large storage magazines at the mine. Orders for explosives are issued two or three days in advance of when needed. The powder is transported in special explosive cars 13 miles to the mine where it is directly routed out on the "hill." The general drilling and blasting foreman ascertains what locations are in need of powder; the car is then diverted to the various places where powder is deposited in 200-case capacity portable steel magazines, one or sometimes two of which are located on each level. At present 1073 50-lb cases of powder are delivered twice a week to the mine. Adjacent to the portable magazines is another portable sheet-iron house which is used for a combined fuse storage, equipment storage and eating house for the men in the drilling and blasting department. Specially constructed wood-lined cupboards in these shacks receive the 6-ft and 2½-ft fuses which are cut and capped in the drill repair shop. Both the eating shack and the powder magazine are portable to allow movement by railroad crane after track alignment. Locks are placed on both the powder magazines and fuse cupboards, keys to these locks being issued to powder foremen only. A daily record of powder used, received, and on hand for each magazine is maintained by the level drill foremen. The powder is transported from magazine to the point of use on a railroad push car, one of which is on every level, or on empty ore or waste cars. Fuses are carried in a canvas sack to the place where the primer cartridges are made up near the springing and loading operations. The primer cartridges then are stored in wood-lined boxes waiting for use. All unused powder and caps are replaced in the magazines at the end of shift.

Springing and Loading

Springing and loading operations have not experienced radical changes over a period of years as have the types of drilling equipment and drilling methods. Springing of holes is probably a little heavier now. Shovels can back up more easily or bulldozers are available now to uncover holes that may be buried by heavy springing. Buried holes were formerly uncovered by hand. The average springing now utilizes 5 springs, with 60 sticks of Gelamite No. 1 on the final spring. Water for stemming is applied in the proportion of 1 gal per 10 sticks. With this springing, the chamber is capable of receiving 250 lb of powder. Exceptional areas have been sprung as high as 10 times with 130 to 140 sticks of powder on the final spring, giving room for 350 lb of powder per hole. Insertion of the cartridges into the chamber is still made through a 1½-in. id iron pipe, 24 ft in length. The primer cartridge is then "spit," tamped in by a 26 ft tamping stick, and the stick and loading pipe then are withdrawn from the hole. Conversion of the water to steam by the detonated charge, carries the rock particles and dust from the chamber in a geyser effect, leaving a clean chamber. Following the final spring, the hole is loaded.

When the main bulk of the powder is in the chamber, 30 ft of Primacord is laced, then taped into a primer cartridge. This is inserted into the chamber followed by

several booster cartridges; the loading pipe is withdrawn and the collar of the hole plugged with the heavy wax paper from the powder case to indicate the hole is loaded. Springing then proceeds to the next hole. Any number of holes may be shot at one time. Preparatory to blasting, the mainline Primacord (Fig 2) is laid along the toe of the bank in front of the holes to be shot. The branch-line cords are then tied on in the conventional manner. Very seldom is there any blasting before 3:45 p.m.; or 45 min after shovel and train operations cease. The only blasting done during regular operating hours is that required for occasional downholes or boulders in the shovel pit. Warning is given before impending shots by whistle at the location of blasting, which, in turn, is amplified to a general warning by a central whistle plainly heard over the entire hill. Powdermen are stationed at various points surrounding the blast to prevent workmen from proceeding into the blasting area. Upon receipt of signals from everyone concerned, four long blasts are blown on the whistle, the double-primed fuse is "spit," and the workmen retire to a safe distance. Cessation of blasting operations within a certain locale is indicated by two long blasts on the whistle.

The use of group-hole blasting with Primacord results in increased ground breakage efficiency and at the same time insures maximum safety to workmen. Networks of high-voltage power lines encircle the property and extend across the various levels, with possibilities of strong electric ground currents. Under such conditions the use of electric-blasting circuits for group-hole blasting would present a problem not confronting the user of Primacord.

Single-hole blasting, using safety fuse for detonation, continues to be used in mine areas immediately adjacent to and above the town of Bingham and where levels directly above blasting areas are below normal widths.

Downholes

At the completion of a clean-up cut, and after the track has been lined over (a distance recently increased to 28 ft for the full-revolving shovel), the bank should be in readiness for the digging cut. Downholes of 24 ft are drilled ahead of the shovel with an average spacing of 27 ft, and back from the edge far enough to put 17 ft of burden horizontal from the bottom of the hole. A slight pitch of the hole away from the vertical towards the edge of the bank aids greatly in flattening the bank when blasted. These holes are sprung lighter than toe holes, the average springing consists of three springs with 20 sticks used in the final spring. Water for stemming is used more sparingly than on toe holes, and in some localities, it is found to be more advantageous to use fine sand for stemming or even no stemming at all. Hercomite Bag powder is being used more extensively every day, the use of Gelamite No. 1 is confined mostly to wet holes. The ease and speed of loading holes with the free-flowing powder makes it a favorite with the powdermen, plus the added fact that additional breakage is the result of its usage.

Free-flowing Powder Loading in Toe Holes

Additional breakage and fragmentation gained through the use of slower acting powder in downholes, led the drilling and blasting department to the belief that the Hercomite Bag powder might prove equally effective in toe holes. The problem then presented itself of conveying loose powder to the blasting chamber, a distance of approximately 24 ft through a drill hole, dipping only slightly from the horizontal. Experimental use of pneumatic powder-loading machine proved most satisfactory and four of these machines are now in use.

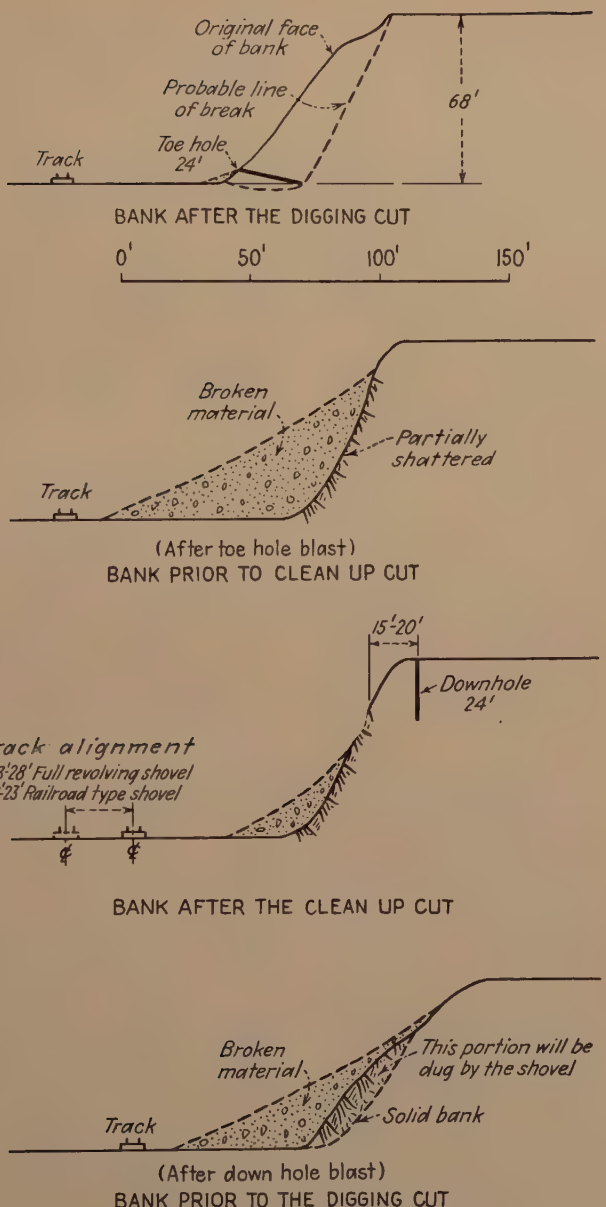

FIG 2—PRIMACORD BLASTING.

Major advantages gained through the use of the pneumatic loader are:

1. *Safety*—elimination of two men working continually under the bank. It is in the springing and loading operations where most accidents occur.

2. *Backbreak*—the slower powder which is loaded into the hole results in additional fragmentation up and into the bank, thereby reducing the number of bank holes necessary.

3. *Damage*—Hercomite reduces the risk of damage to electrification and equipment by its non-throwing, slow-heaving action.

4. *Manpower*—a springing and loading crew normally consists of three men. If the holes are previously sprung, a two-man crew can load holes quicker and easier with this method than a three-man crew can load with cartridges.

5. *Costs*—blasting costs are lessened as a result of all of the above named reasons.

Trimming

Safety under the banks for powdermen and shovels is attained through careful bank trimming. It is in this branch of the work that calm, self-reliant workmen are a necessity. Loose rock material which is left hanging above the reach of the shovel dipper must be removed by physical effort or by explosives. Bank trimming is accomplished by powdermen who descend the banks on ropes at the end of the shovel-operating shift, bar down small loose material and dig "pot holes" adjacent to large boulders or slabs. "Bunches" of powder, which are 15 to 20 cartridges tied with the 6-ft fuse, are then put in the "pot holes," fuses are "spit," and the men ascend the bank to safety. Following the detonation of the charges, the men return to continue the trimming of small material and again repeat the "pot holing" if necessary.

Certain extremely high banks of past years have been reconstructed into two or three separate benches or levels. The elimination of these high banks, together with the installation of larger full-revolving shovels, and the increased use of downholes and Primacord, has resulted in a substantial reduction in the amount of bank trimming necessary. Bank trimming remains, however, an important and integral part of the duties within this department.

Churn Drilling

During 1937 and 1938, blast-hole drilling was extensively tested against the conventional piston-drill toe hole method in use at that time. A Bucyrus-Erie 29-T electric drill with a 9-in. bit was used. Hole spacing averaged 21 ft with 30 ft of burden on the toe at a depth of 58 ft, or 8 ft below grade. Drilling speed was 7 ft per hour or one hole per shift, which included springing and loading. Two springs per hole provided sufficient chamber to load with 500 lb of explosives, which resulted in breakage of 2.2 cu yd per pound of powder used.

A comparison of the two methods disclosed the disadvantages of blast-hole drilling: (1) cubic yardage per pound of powder was only 2.2 compared with 4.4 for the toe-hole method; (2) cubic yardage per man-shift was slightly lower, being 365 yd as against 369 yd; (3) the problem of maintaining water-supply lines to the drills in the winter months was tremendous; (4) lack of level room in certain areas of the mine precluded the extensive use of this method; (5) certain amounts of supplementary toe-hole drilling proved necessary to break the hard toe which often remained after blasting; (6) overall operating costs per cubic yard of material broken was practically double.

The advantages of blast-hole drilling are: (1) the elimination of bank trimming; (2) damage to electric shovels is lessened by avoiding the digging cut; (3) safety for the workmen is increased by not working under the bank; (4) greater horizontal distance for each lining of the track.

These comparative tests were made when the old type piston drills were still in use.

Comparison of results to be obtained as between churn-drill and present type hammer-drill operations would prove more favorable to present drilling methods than what was obtained in previous tests.

It is only in old dumps which are being excavated that churn drilling has been of practical value. This fill material has been recemented, yet is of such character that the shovel is able to dig the toe of the bank without blasting. Excavation of the toe leaves a treacherous overhang which often resulted in shovel coverage. Bucyrus-Erie 29-T churn drills have relieved this condition by drilling holes 30 ft in depth. Holes are carried with two casings, the outer one 12½ in. by 14 ft and the inner one 10½ in. by 26 ft, the last 4 ft are not cased. Holes are immediately loaded with 150 to 200 lb of explosives and Primacord inserted into a 30-ft pipe which is lowered to the loaded chamber. Casings are then removed. Tamping in the form of fill material is barred loose from the collar of the hole, and the pipe is withdrawn, stringing the Primacord through the stemming as it is raised. After blasting the upper portion of the bank is sloped sufficiently to permit safe shovel operations.

Conclusions

In studying the comparative figures set forth in Tables 1 and 2 certain explanations are necessary for a clear concept of the results obtained.

Shovel shifts—this is a variable figure which may change several times during the year.

Cubic yards loaded per shovel shift—the increase in yardage loaded per shovel-shift is primarily because of larger shovels and more flexible train operations.

Cubic yards per drilling and blasting manshifts—(1) greatly increased drilling speeds of the DA-35 over the old piston drill; (2) substitution of downholes for bank holes in bank sloping; (3) increase in the depth of the holes by 2 ft; (4) use of motor cars and push cars for moving equipment; (5) use of Primacord. Concerted effort of many explosive charges, detonated simultaneously, increased fragmentation throughout the bank; (6) a considerable reduction in the amount of secondary blasting and bank trimming required; (7) greater breakage per loaded hole resulting from heavier charges. The added reach of the new type shovel affords more room for the blasted material to occupy; (8) substitution of mechanical methods (bulldozers and shovels) for hand methods in uncovering buried holes. Occasionally, drill grades are partially prepared by bulldozers.

Table 1—*Comparative Figures*

Operation	1927	1937	1946
Average shovel shifts per day	19.8	34.1	21.5
Cubic yards loaded per shovel shift	1,856	1,991	2,773
Cubic yards per drilling and blasting man shift	253.5	368.9	742.5
Number of man shifts per shovel shift (includes supervision)	7.2	5.4	3.7
Drilling machines operating per shovel shift			0.6
Footage per drilling man shift	15–20	15–20	41.3
Footage drilled per shovel shift	61.8	58.4	55.6
Footage drilled per 1000 cu yd	39.3	33.6	20.0
Holes sprung and loaded per loading man shift	2.7	2.6	2.3
Cubic yards per pound of powder (includes trimming and secondary blasting)	4.5	4.4	5.6
Pounds powder per 24-ft hole	180	200	237
Percentage of man shifts—drilling	57 %	55 %	52 %
Percentage of man shifts—trimming banks	18 %	18 %	9 %
Percentage of man shifts—springing and loading	23 %	24 %	35 %
Percentage of man shifts—2 in. air-line repairs	2 %	3 %	4 %
Cost per 24-ft hole (includes labor, material and powder)	$39.36	$46.83	$56.85
Overall cost per 1000 yd—drilling and blasting	$54.68	$46.84	$43.08
Percentage of total mine operating costs charged to drilling and blasting	23.9 %	22.5 %	16.9 %

Holes sprung and loaded per loading manshift—this shows the least change of any figure. Increased usage of the pneumatic loading machine is expected to raise this figure within the next year. An estimated 25 to 50 pct increase in loading per manshift seems to be the result possible with this machine.

Cubic yards per pound of powder—better yardage is partially the result of the extensive use of Primacord, less bank trimming and less secondary blasting.

Cost per 24-ft hole—cost of labor, material, compressed air, Primacord and other expense items have steadily increased over the years. Although the 24 ft hole is costlier, increased yardage per hole blasted justifies its selection.

TABLE 2—*Average Monthly Data*—1946

Consumption of Explosive Supplies	
Pounds of explosives	327,658
Feet of safety fuse	41,992
Number of 6-X blasting caps	12,428
Lineal feet of Primacord	71,000
Drilling Supplies	
Sharp bits issued	4,720
Lineal feet of new drill rods issued	981
Lineal feet of 1-in. air hose issued	258
DA-35 machines located on the levels	67
DA-35 machines held in reserve	8
X-71 machines in use	7
Jackhammers available	18
Drilling Records	
Drilling man shifts	883
Total footage drilled	36,469
Footage per man shift—toe holes	40.6
Footage per man shift—bank holes	25.6
Footage per man shift—downholes	48.5
Average footage of churn-drill holes per month	1,165
Springing and Loading Records	
Springing and loading man shifts	608
Number of holes sprung and loaded	1,381
Man shifts trimming banks	155

Continual lowering of costs and increasing worker and machine efficiencies, are constantly being held in mind. Determined efforts to seek the most efficient and economical methods of preparing rock for excavation have been amply rewarded, as shown in the final comparison.

ACKNOWLEDGMENTS

The author wishes to extend his thanks to Louis Buchman, General Superintendent of Operations, for his permission to prepare this paper; to H. L. Garrity, Superintendent of Mines, L. F. Pett, Assistant Mine Superintendent, and Albert Kastelic, General Drilling and Blasting Foreman, for their full cooperation and guidance; and to Pete Arata, Pete Petraca, and W. W. Raby, Assistant Foremen, for an abundance of information pertaining to Drilling and Blasting operations.

Laying Panel Track at the Morenci Open Pit

By Walter C. Lawson,* Member AIME

The primary objective in laying track in panel sections is to reduce the number of track laborers required. This is possible because the work is mechanized. Moreover, because the work is mechanized and each of the operations needs only a few men, and each is self-contained, the work can be carried on at night as well as in daylight; therefore, the method provides a means of preparing tracks on more than one shift.

The laying of tracks in open pits and quarries in panel sections is not new but new methods have been made possible by the introduction of new types of equipment. The purpose of this paper is to describe the methods that are followed in the Morenci open pit.

General Mining Operations

Full-scale ore production at Morenci is about 50,000 tons daily. The normal yearly output of ore and waste is 32,000,000 tons, of which 30,000,000 tons is handled by rail haulage. This requires the building of approximately 47 miles of loading and other temporary tracks during a 12 months' period (Fig 1).

Broken ore and waste is loaded into 90-ton capacity dump cars with 5-cu-yd full-revolving electric shovels. Shovel banks are uniformly 50 ft high. They are blasted with churn drill holes and the broken material from a blast is loaded out with two shovel cuts, the first one being called (locally) the "splatter" cut and the second one the "clean-up" or "face-up" cut. When a shovel is making its clean-up cut, the loading track is about 70 ft from the toe of the bank and, under normal conditions, a bank is blasted against it without removing the track. It is thus evident that each track is used for two shovel cuts before relocation is necessary, inasmuch as a single position serves for both the clean-up cut prior to blasting and for the splatter cut following the blast. The normal advance of a solid bench by a single blast is 40 ft (Fig 2).

Preparation of Panel Grade

In preparation of panel grades bulldozers and rooters are used for the bulk of the work and an auto-patrol road grader for the final stage. At Morenci, bulldozers have always been used for track-grade preparation, but before the introduction of the other two types of equipment grades were uneven and irregular at best, and always necessitated a big gang of track laborers to hand-block under the ties with rocks after a track was laid into place. High blocking also made poor track because of instability and caused frequent derailments. Under these conditions derailments were generally bad as re-railing became difficult. Much track was torn up and many ties broken in the process.

The inadequacy of the hand-blocked track, together with the inefficiency of hand blocking, constantly pointed to the desirability of reducing the amount of labor required, so various means were tried to improve conditions, such as wood

Presented at the Arizona Section Meeting, Tucson, Oct. 28–30, 1946. Manuscript received at the office of the Institute Nov. 12, 1946. Issued as TP 2189 in Mining Technology, July 1947.

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wedges instead of rocks and a clamshell bucket mounted on a motor crane to place fine material near the track for ballast. None of the methods tried proved to be effective and it became more and more evident that better results could be attained only by the development of a procedure whereby a uniform grade could be prepared by machines from the material at hand. Accordingly, by gradual steps, rooters and road graders were introduced into the operations. The purpose of the rooter is to tear out hard bottom, which at the same time makes enough fine material for use in the grade. It is surprising to one who is not familiar with the use of rooters to see the tight character of rock that can be broken up with them. The purpose of the road grader is to give the grade a final leveling, so that no hand blocking of track is required.

Fig 1—Morenci open pit.

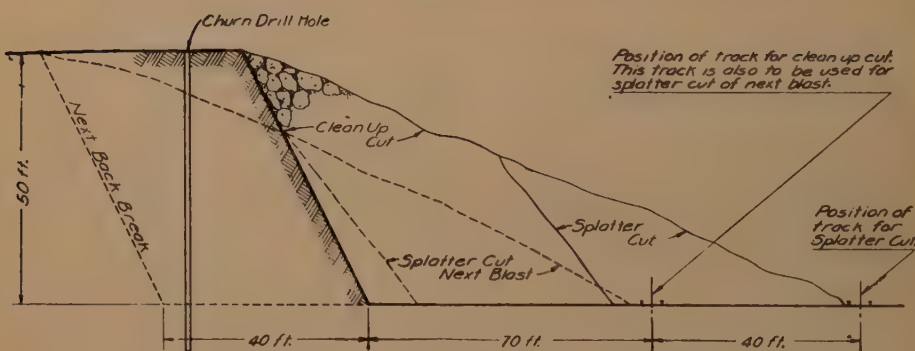

Fig 2—Typical diagram of mining bench and track positions.

As soon as a shovel has completed its splatter cut on a blast, it becomes necessary to relocate the track near the remaining broken material so that it can be loaded. When a shovel has progressed

FIG 3—BULLDOZER MAKING FIRST PASS ON PANEL GRADE.

FIG 4—USING ROOTER TO BREAK HARD BOTTOM.
Under severe conditions a pusher bulldozer is used as shown in photograph.

FIG 5—BULLDOZER GRADE PRACTICALLY COMPLETED.
An auto patrol road grader will follow to complete the leveling.

FIG 6—FINISHING GRADE WITH ROAD GRADER.
This eliminates hand-blocking of track.

along the bench sufficiently far to permit working room, general practice calls for the assignment of a bulldozer in the preparation of a panel grade for the laying possible with a bulldozer, but although such a grade is relatively smooth and has a good appearance, further leveling with a machine designed for that purpose

FIG 7—LOCOMOTIVE CRANE AND THREE FLAT CARS OF PANELS BEING MOVED OVER 4.0 PER CENT SERVICE SWITCHBACK.

of a new loading track (Fig 3). The roughing out of the panel grade is accomplished entirely by bulldozers with the use of rooters. On approximately half of the benches in the Morenci mine the material breaks so that a bulldozer can establish a grade without the use of a rooter. On the waste benches, which are harder, and also on some ore benches, it is necessary to use a rooter. At times, under severe conditions, one bulldozer pulls the rooter and a second one pushes it (Fig 4). Only on rare occasions is it necessary to drill and blast the high humps that extend above the shovel grade. It is always necessary for the bulldozer operator, by sorting, to make suitable use of the coarse and fine material (Fig 5). It is the practice to make the best grade is needed to save a considerable amount of labor that otherwise would be required to hand-block the ties after the track is laid. It has been found that under average conditions four bulldozer shifts are required to prepare 1000 ft of panel-track grade.

Upon completion of the "rough" grade by a bulldozer, a road grader is used for the final preparation prior to laying of the track panels. An auto patrol with a skilled operator can prepare 4000 ft of good bulldozer grade in a shift (Fig 6).

LAYING PANELS

After the grade is prepared as described, it is ready for the laying of panels. This operation is under the direction of a panel-track foreman. He makes the necessary arrangements for the required number

of panels from the panel yard, and they are delivered on flat cars to the mining benches by a 600-hp diesel locomotive (Fig 7) according to a schedule. The panel-track work train consists of three, four or five flat cars loaded with panels, a locomotive crane and a locomotive. Inasmuch as ruling track grades on the switchbacks connecting the benches of the pit are 4.0 pct, it is impossible for a locomotive crane to move itself and loaded flat cars from the pit benches to the panel-building yard; therefore, a locomotive is assigned for this purpose.

A standard panel consists of two 90-lb rails each 33 ft long, fastened with the ends exactly even to seventeen 7 by 9-in. by 8-ft cross ties all equipped with tie plates. Railroad spikes, 5/8 by 6-in. size, are used to hold the rails to the ties. When panels are built the ties are spaced so that at the time of laying a tie can be slipped under the rail joints. These are called "joint ties" and are required as standard practice. They are inserted after the panels have been laid and the rails are spiked to them (Fig 8).

FIG 8—SLIPPING JOINT TIE UNDER JOINT BARS.

The locomotive crane used in panel laying is operated by a diesel engine of 40 ton capacity, equipped with a 50-ft boom. Panels can be handled over a distance of 50 ft from the center of the crane. The crane weighs 81 tons, is self-propelling, is equipped with standard couplers and air brakes, and can move itself and two loaded flat cars on level track. Also in use in the mine are one 25-ton and one 50-ton diesel-locomotive crane. The small size can be used successfully in the panel-track work but the 40 or 50-ton sizes are preferred because they are heavy enough to provide stability when swinging panels with a low boom.

When laying panels, the locomotive crane works from the "old" track that was used as a loading track for the clean-up cut by the shovel. Working from this old track, panels are lifted from a flat

car and laid into place on the grade that has been prepared (Fig 9). A special set of two rail clamps and short sling is used for lifting the panels and each length of the bolt-hole spacing and requires slipping the joint bar so that the end holes of the bars catch the two end holes of the rail (Fig 11). It has been found

Fig 9—Locomotive crane swinging panel into place on finished grade.

panel is held by the crane operator until the joint bars have been applied by the ground men. It is possible in this manner for the operator to swing or adjust the position of an ingoing panel to facilitate joining up of the bolt holes in the joint bars (Fig 10).

Panels being laid on curve alignment are handled in exactly the same manner except that at the time of joining the panels a "dutchman" (short piece of rail) is inserted between the two rails on the outside of the curve. Dutchmen have been standardized as to two lengths in multiples of rail-hole spacing so that the rails do not require additional bolt-hole drilling. The smaller of the two sizes is the same as the bolt-hole spacing; therefore the joint bar slips one set of holes; whereas the longer of the two lengths is twice the that these two sizes of dutchmen permit the laying of any curvature required in the mine, the number and size to be used being determined by the panel-laying leadman as work progresses in the field.

The standard panel-laying crew consists of one Brownhoist operator, one panel-laying leadman, and three panel tracklayers. Two such crews are used daily, one being assigned to the day shift and the other to a night shift. One crew can lay an average of 40 panels in a shift and under favorable conditions as many as 50.

After a "new" piece of track has been laid it is then necessary to hook it up on both ends with existing old track, and also to ballast it. The hook-ups are made by disconnecting the old track at the appropriate place and throwing that

FIG 10—PANEL HELD BY CRANE OPERATOR TO FACILITATE MAKING JOINT-BAR CONNECTIONS.

FIG 11—COMPLETED PANEL JOINT ON CURVED TRACK.
Note dutchman in far joint and that end bolt holes in joint bar catch end holes in rails.

FIG 12—TRACKSHIFTER HAS JUST COMPLETED THROWING OLD TRACK OVER TO CONNECT WITH NEW PANELS.

FIG 13—TRACKSHIFTER PULLING OLD TRACK OUT OF SHOVEL OVERCAST.

section of track over with a track-shifter so as to meet the new section, (Fig 12). This part of the work is always done on the day shift with a small track gang.

Switches for use in bench-track operations are also laid with a locomotive crane. They are prefabricated in a single section in the same way as a panel, but longer, of

FIG 14—PLACING SHORT, PREFABRICATED SWITCH WITH CRANE. NOTE HANDLING SLING.

Panels are laid so as to anticipate favorable connections, in order to minimize the amount of rail cutting required and, as a matter of practice, standard-length rails are never cut but connections are made by using short panels and short pieces of rail.

Immediately before the old track is disconnected and thrown over to join up with the new panel section, a track-shifter is used to pull it out of the overcast that has accumulated from the shovel-loading operations (Fig 13). After the track has been raised so that the ties are clear of all rocks, the joint bars are unfastened by removing one track bolt and loosening the other one. All loose material is also removed from the top of the ties and the used panels are then ready to be picked up.

course. These are locally known as "short switches." They are built by using a single 33-ft lead rail between the heel of the switch point and the frog, making a total over-all length of approximately 60 ft. They are handled with a specially designed lifting sling. Two flat cars hooked together are used in transporting them. Normally the time required to unload a prefabricated switch from the flat cars and put it into place, together with the necessary hook-up to adjoining tracks, is about one hour. The switch stand itself is spiked into place after the switch has been laid (Fig 14).

Under some unusual conditions it is necessary to lay panels "end-over-end," which is the local term for the crane to be working off the same track it is building. This offers no real obstacles but is avoided

where possible, because the method is slower and does not permit of any flexibility, inasmuch as a bench is not usable or even passable until the track is completed and hooked up.

Picking Up Panels

The haulage locomotives (diesel electric and combination trolley and storage battery) in use at Morenci weigh 125 tons. The dump cars weigh 45 tons, carry an average payload of 90 tons, making a total gross weight of 135 tons on eight wheels. This is very high wheel loading. Standard trains consist of eight cars and a locomotive. The high wheel loading demands good loading tracks to prevent excessive derailing, and for this reason it has been found in the Morenci operations that it is more economical to pick up panels after every use, load them on flat cars and send them to a permanent rebuilding yard rather than to attempt to relay them on the mining benches without thorough rebuilding. Also, in view of the use of the old tracks to work from when laying new panels, it is apparent that the picking up of old panels and rebuilding them follow as a natural step.

Picking up of used panels is simply a reversal of the laying procedure, with the crane and flat cars working off the new section of track that has just been completed adjoining and parallel to the one being removed.

Ballasting

Inasmuch as the goal in the preparation of the panel grades is to obtain a uniform grade, not much ballasting is involved. On curved track the auto patrol builds the superelevation into the grade as required.

The bulk of the ballasting needed is accomplished by a bulldozer. In the preparation of the panel grade a windrow of fine material is left on one side of the grade and, following the laying of the panels, a bulldozer can quickly spread the material over the ends of the ties and under the rail on that side. This procedure successfully holds the track in place and prevents it from kicking out sidewise. A little fine material is put against each side of the ties under the rail on the opposite side of the track with hand shovels. As ties are seldom more than ½ in. from the grade when first laid, all make grade contact after being run over once (Fig 15).

Panel Building and Reconditioning

New panels are made up in the yard especially designed and equipped for rebuilding panels. All materials needed, such as ties, rails, spikes and joint bars, are stored near this yard. A jig, which is made of concrete to determine tie spacing and of vertical uprights to determine rail gauge, is used when new panels are being built. All panels, whether for curved or tangent track, are built to standard gauge. The two rails are laid on top of the ties with a diesel-locomotive crane, tie plates are inserted and spikes are driven with a pneumatic spike driver. Angle bars are attached in a manner already described, and ordinarily only two track bolts are used per joint, one for each rail. Upon the completion of the work on the ground, the crane picks up the completed panel by the use of a set of rail clamps, and either loads it on a flat car for transportation into the pit or puts it in a storage pile, depending upon immediate requirements.

The practice in rebuilding or reconditioning is similar to that of building new panels, except that the used panels are placed on different types of jigs, which are simply a pair of rails laid level in concrete and projecting about 1 in. above the yard level. In rebuilding, ties are replaced if needed; rails, also, if they have been bent; gauge is checked and loosened spikes are reset. A locally made

rail straightener is used to straighten bent rails.

Under normal conditions approximately 200 panels are maintained in storage in the panel yard all ready for use in the pit. This provides a good reservoir of track that can be drawn on to meet routine as well as unusual demands. The latter occurs at times because several shovels cutout at the same time or because it is necessary to relocate long passing tracks.

Fig 15—Track at left has just been completed.
Note ballast over ends of ties on one side of track. Track at right will be picked up by crane working off track at left.

The panel yard is operated on a two-shift basis, the normal crew per shift consisting of one crane operator, one panel-building leadman and four panel-building laborers. A crew can build 12 new panels per shift or rebuild 30 panels per shift.

Advantages, Costs and Organization of Work

The greatest use of panel track is in building shovel-loading tracks where the useful life is only a few weeks; however, the practice has been extended to include passing tracks and some other running tracks that may be used for several months. In these instances a bulldozer covers the ends of the ties with ballast on both sides of the track. Frequently dump tracks are also laid in panel sections. Main-line running tracks of a permanent nature are laid with staggered joints and hand-tamped with slag ballast.

The fact that panel-track operations can be conducted on more than one shift is of major importance at the Morenci mine, because of the high rate of mining per foot of working bench. The rate currently is 3 tons per day per foot of bench exposed, and earlier in the operations the rate on some of the benches was three times as great. Another important advantage is that the panel-track crew provides good flexibility with respect to other operations where a locomotive

crane is needed intermittently, but frequently on work such as placing power poles for trolley haulage, handling materials, and handling heavy parts for shovel repairs.

Trolleys and trolley structures are not needed on the loading benches at Morenci because of the use of storage-battery auxiliaries on the electric locomotives. It is believed that side-arm collection like that ordinarily used when benches are electrified would not offer any interference in laying panels but would require the picking up of the old panels prior to the time the trolley wire was erected beside the new track.

The costs of building panel tracks as distinct from main-line track maintenance, new track construction and extensions, have not been kept at Morenci. In any event, such figures probably would not be of interest for comparison with other mining operations because of differences in labor supply, types and size of haulage equipment, rate of mining, and other factors. It can be said, however, that during the extreme shortage of common labor during the World War II period, it would have been impossible to approach the production attained without the use of the methods that have been described.

Of interest, perhaps, are the costs of operating the mechanical equipment. For the year 1945, per shift, including the operator's time, costs were as follows:

Bulldozers and road graders......	$35.76
Diesel locomotives...............	40.82
Locomotive cranes................	20.21
Trackshifters...............	about 20.00

During this period the rate of pay for locomotive crane, bulldozer and road-grader operators was $8.16 per day; for locomotive engineers, $8.21; for track-shifter operators, $7.52. Operations were conducted on the basis of a 48-hr week; therefore the costs given included premium pay for all hours worked in excess of 40.

Also of interest may be the organization of the work. In the Morenci operations the work of grade preparation and the panel work are divided between two departments, each under the direction of its own foreman. The first comes under the bulldozer foreman and the second under the panel-track foreman. Neither has any responsibility as to the work of the other and both are responsible to the General Mine Foreman, who is in direct charge of all the work.

Progress of Mining Studies at Bureau of Mines Oil-shale Mine, Anvil Points, Rifle, Colorado

By E. D. Gardner* Member AIME

(New York Meeting, March 1947)

Introduction

Oil shale deposits have been exploited in various countries throughout the world, but generally with government aid. The oil-shale industry of Scotland perhaps is the oldest and best-known; the largest was conducted in Manchuria by the Japanese during World War II as a source of liquid fuels for military uses. In general, foreign oil-shale deposits that have been exploited are much lower grade than those in the western United States. Considerable interest began to be shown in western oil shales in 1916; the peak of activities was in 1923. The Catlin operation, near Elko, Nevada, perhaps was the nearest approach in this country to an oil-shale enterprise on a commercial scale. Over 100 companies were formed, ostensibly for exploiting the oil shale; a large number of them, however, proved to be stock promotions only, which gave the embryo industry a black eye. The Bureau of Mines operated an experimental oil-shale retort plant near Rifle, Colorado, from September 1926 to June 1927 and again from April 1928 to July 1929, after which the plant was dismantled. By this time the East Texas oil fields had come in and the country lost interest in substitute liquid fuels.

During World War II the Nation again became concerned about the magnitude of its petroleum reserves. Congress (April 1944) directed the Bureau of Mines to build and operate demonstration plants to produce synthetic liquid fuels from coal, oil shales, and agricultural and other products; a five-year program was authorized. As a part of this program the Bureau of Mines has built, at Anvil Points near Rifle, Colorado, a retort plant and developed an experimental mine to supply the plant with oil shale. An important part of the mining work at Rifle is to devise methods and select practices for mining the oil shale on a commercial scale at the lowest practicable cost. The product to be mined has a relatively low value, a fact that must be kept continuously in mind. The combined aim of those in charge of the mining and those in charge of retorting is to demonstrate that the shale oil can be produced at a cost per barrel corresponding to the current quotation for crude petroleum—or at least very close to it. Plans call for an uninterrupted flow of broken oil shale from the face in the mine, through crushers, through the retorts and thence to the disposal dump. The disposal of the spent shale in the Rifle area should present no unusual problem.

Mining research has consisted so far of paper studies of the general phases of the problem, determination of the widest span that the roof stone will stand safely unsupported, and drilling problems. Underground development has not proceeded far

Published by permission of the Director, Bureau of Mines, U.S. Department of the Interior. Manuscript received at the office of the Institute March 25, 1947. Issued as TP 2286 in Mining Technology, November 1947.
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enough to do experimental blasting in stopes or to obtain operating data on loading or transportation of the broken shale. In general, standard mining practices will be followed as far as practicable, but modifications will be made to fit conditions at the oil-shale mine. Experimental work is being done to reduce the cost of individual phases of mining, which appears too high; some success has been attained.

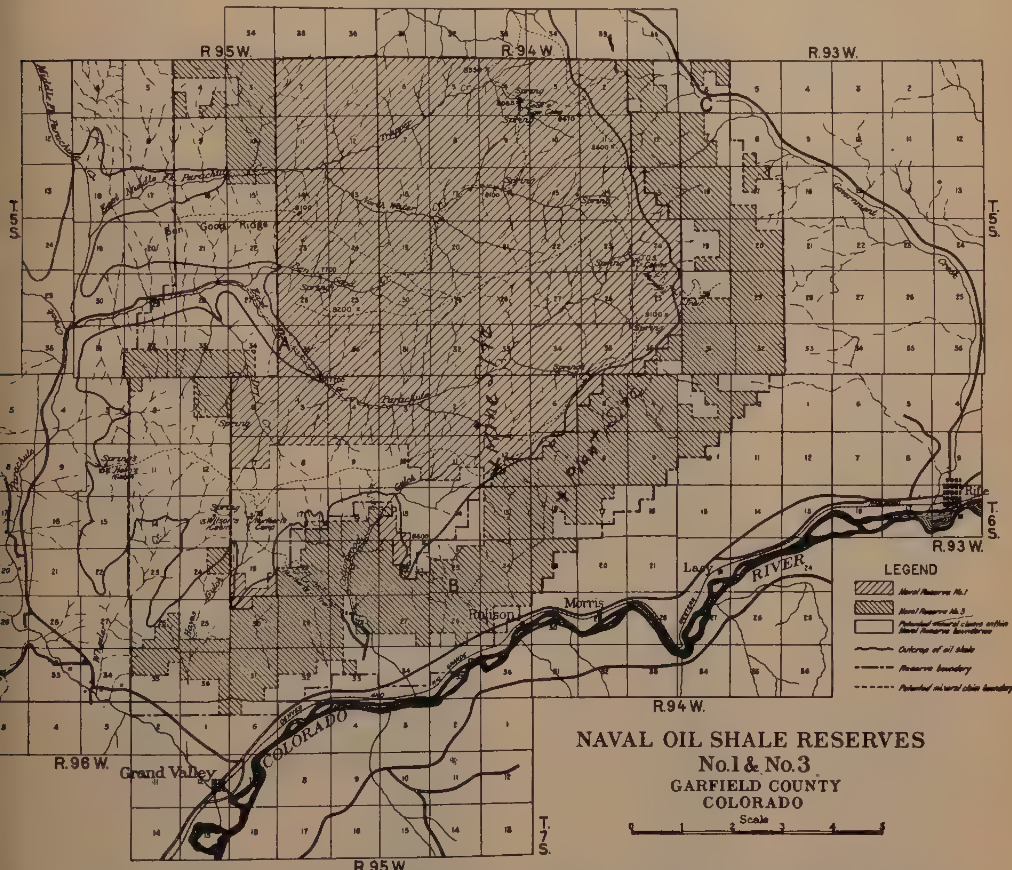

Fig 1—Naval oil shale reserves no. 1 and no. 3, Garfield County, Colorado.
(Martin J. Gavin and John S. Desmond: Construction and Operation of Bureau of Mines Oil-Shale Plant 1925–1927. Bur. of Mines *Bull.* 315 (1930) 12).

Deposits

The most important oil-shale deposits within the United States occur in the Green River formation in Colorado, Utah and Wyoming. A total of about 7000 square miles in this area has been classified as being chiefly valuable for the contained oil shale. The oil shale of western Colorado generally is more amenable to exploitation, apparently richer and probably more persistent than elsewhere in the Rocky Mountain region. The current work being done by the Bureau of Mines is on Naval Oil Shale Reserves No. 1 and No. 3 near Rifle, Colorado. The mine site (Fig 1) is about 5½ miles by a mountain road (10 pct grade) from the plant site, which is in turn 2 miles

from U. S. Highway 6 and 10 miles from Rifle, Colorado. The highway parallels the Colorado River, the Denver and Rio broken down and volatilized by the application of moderate heat; the condensate is the shale oil, which looks like petroleum.

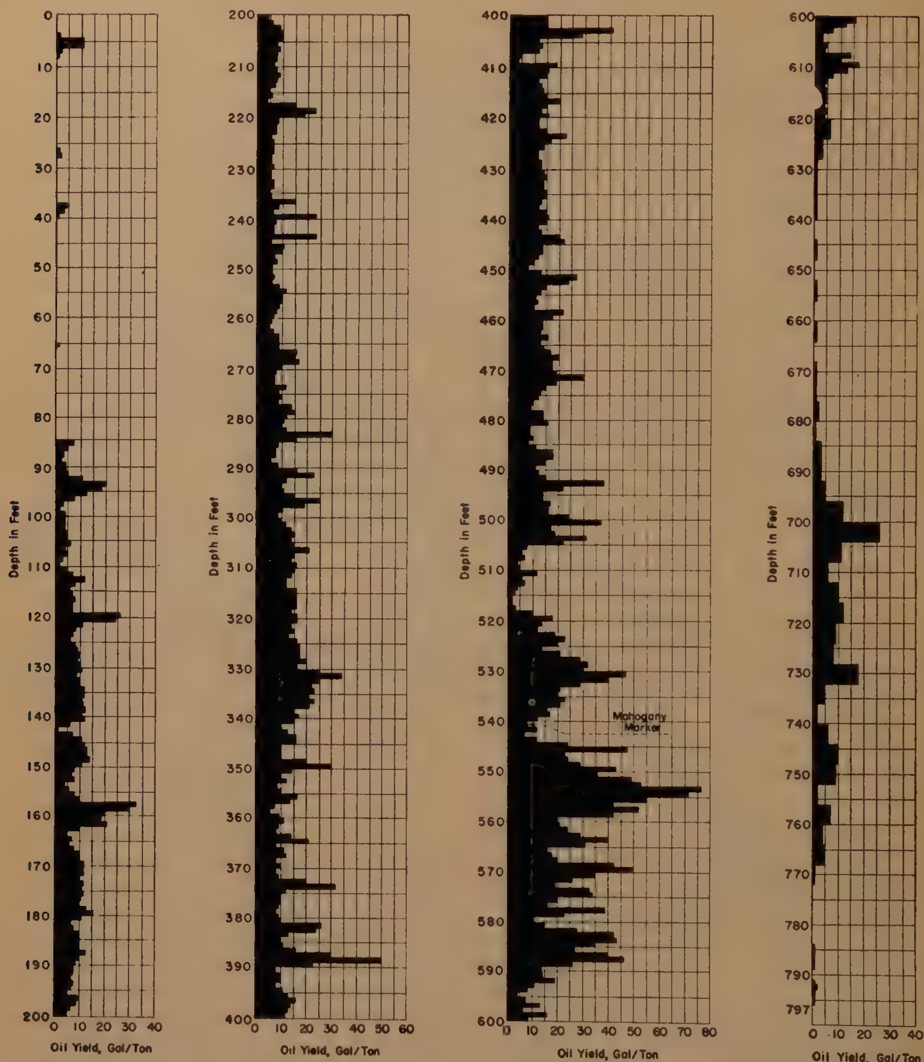

FIG 2—LOG OF GREEN RIVER OIL-SHALE BEDS AT MINE OF OIL-SHALE DEMONSTRATION PLANT, RIFLE, COLORADO.

Test hole *A*, U. S. Naval Oil Shale Reserve No. 1, Garfield County, Sec 12, T6S, R95W.

Grande Railroad, and a 66,000-volt public utility power line.

The oil shale of the Green River formation contains organic matter but no free oil or free carbon. The organic matter is

The shale beds on Naval Oil Shale Reserve No. 1 (Fig 1) comprise a tough, strong rock. "Shale" really is a misnomer, as the rock is a marlstone and has few of the qualities usually attributed to shale. The oil shale

outcrops at the base of a long line of cliffs about 3000 ft vertically above the valley floor. The formation at the mine site has a 5° dip; it flattens to the westward. It is not cut by faults and as far as is known has no local changes of dip and strike; it has relatively few jointing planes or vertical planes of weakness. Some of the bedding planes constitute definite planes of weakness, but they are less evident as depth is attained. The specific gravity of the oil shale is relatively low; the average at the mine site is 16 cu ft to the ton. About 6000 tons of oil shale was mined by the Bureau of Mines in its first operation, four miles from the present mine; this work indicated that the oil shale should stand well in open stopes. Underground blasting has indicated that the oil shale tends to break into large fragments.

A log of a core-drill hole (Hole *A*) put down at the mine site is shown in Fig 2. The upper oil-shale measure is 500 ft thick and averages 15 gal of shale oil per ton; the bottom 70 ft averages 29 gal per ton. The oil-shale measure on the Naval Reserve is overlain with barren sedimentary rocks from 0 up to 500 ft thick; the overburden at Hole *A* is 100 ft. A marker bed comprising about 6 in. of volcanic ash in the high-grade zone persists throughout the area

The oil-shale reserves are enormous. The top oil-shale measure of 500 ft thickness contains 850,000,000 tons of shale that could yield 300,000,000 bbl of oil per square mile, as indicated by core drilling at the mine site. The lower 70 ft of the measure contains 130,000,000 tons of oil shale, or a net of 100,000,000 tons of shale that could yield 70,000,000 bbl of oil to the section, allowing for pillars that would be left in mining.

Past experience has shown that the higher-grade layers of oil shale have an unfortunate quality of becoming sticky when processed in a furnace. The 70-ft series of beds has been classified into eight divisions according to the coking qualities of the shale. Current work is being confined to this series, and a mine has been developed to supply the retort with oil shale, on demand, from any of the eight divisions.

Mining Methods

A commercial oil-shale enterprise would comprise a mine, a retorting plant and a refinery. All three phases, of course, would have to be integrated and be of sufficient magnitude that minimum overall costs could be obtained. The scale of operations probably would be governed by the refinery, which we are considering would have a minimum capacity of 10,000 bbl of oil per day, with a 7-day week. An underground mine to match a refinery of this size would have to produce 20,000 tons per day during a 5-day week; an open pit would be about twice this size, as the grade would be about half that in an underground mine. In any event, a mine of either type would have full advantage of size in regard to overall costs.

A study of the log of Hole *A* (Fig 2) shows that the grade of the oil shale varies greatly from foot to foot in the vertical column. The grade, however, persists within the individual beds. Five groupings of beds suggest themselves for consideration in making a choice of the best interval of the column for exploitation (Table 1).

TABLE 1—*Five Groupings of Oil-shale Beds*

Number	Location in Column, Distance from Marker, Ft	Thickness, Ft	Average Grade, Gal per Ton
1	+447 to −53	500	15
2	+263 to −53	315	17
3	+21 to −49	70	29
4	−4 to −49	45	34
5	−9 to −16[a]	8	55

[a] Mahogany bed.

Unless a retort can be designed that minimizes the coking difficulty, the cost of retorting the high-grade beds by themselves would be relatively high. It is expected that this difficulty could be

overcome by retorting a mixture of the high-grade layers with material of medium grade, as in the 70-ft series.

The first decision to be made for exploiting the oil shale commercially on Naval Oil Shale Reserve No. 1 would be between an open-cut and an underground method of mining. The distance from transportation, availability of water, and conveniences for the retorting plant, however, would affect the selection of a site for a commercial mine. It is taken for granted that the retorting plant would have to be located near the mine to save the costs of transporting the oil shale from the mine to the plant.

The 100 ft of overburden at Hole *A* is absent in other areas, and in places the upper part of the 500-ft series has been eroded. Moreover, areas may exist of minable size where relatively little of the formation is left above the 70-ft series. The top 140 ft of the oil-shale measure is relatively low grade, but if removal of this material was necessary to mine the underlying beds it probably would be economical to send it to the retort plant. Present plans do not call for any mining work to be done specifically to establish open-pit costs. Considerable information, however, will be obtained in the underground operations that could be applied to open-pit mining. A pencil study is being made of the general problem, based on geological and topographic maps made by the Geological Survey.

Undercut block caving would come into consideration for mining the oil shale under deep cover; no rock just like the oil shale, however, has been mined. The toughness of the stone and the relative lack of vertical planes of weakness would be handicaps in block caving. I am inclined to believe, however, that a procedure for block caving could be worked out, but it would take more time and more money than are now available for the purpose at the oil shale project. At any rate, work on block caving is being held in abeyance until more pertinent data are at hand.

TABLE 2—*Expected Relative Costs per Barrel of Oil Mined*

Mining Method	Thickness to be Exploited, Ft	Grade, Gallons per Ton	Mining Cost per Ton of Shale	Mining Cost per Bbl of Oil	Total Cost per Bbl with Assumed Retorting Cost of $0.50 per Ton of Shale
Room-and-pillar	8	55	$1.25	$0.89	$1.26
	70	29	0.50	0.71	1.42
Block-caving	315	17	0.40	1.00	2.20
Open-pit	500	15	0.25	0.70	2.10

The bottom 70 ft (which would be mined by underground methods) now appears to constitute the most promising portion of the oil-shale section for commercial exploitation; the research work on retorting is being confined to this series. This part of the measure would have to be included in an open-cut mine for the operation to be commercially feasible; the overlying shale by itself would be too low grade to be exploited in the foreseeable future. Moreover, the existence of open stopes 70 ft high below an open pit would present a mining hazard. Prior mining of the 70-ft series would be less of a handicap in block caving than in open-pit mining. The pillars left in open stopes could be blasted down to provide an undercut area in a caving method.

The 8-ft series, which includes the so-called Mahogany bed, averages 55 gal, or 1.3 bbl of oil per ton. This series alone probably would give the largest money return per ton of shale mined, provided this high-grade material could be retorted alone

as cheaply as when mixed with lower grades.

Table 2 shows the expected relative costs per barrel of oil produced by three methods of mining.

The projected costs per barrel shown in Table 2 are relative and are about the minimum that could be expected in practice. A smaller retorting cost than shown would lower the margin between room-and-pillar and open-pit mining; a higher retorting cost would increase the margin.

Selection of Method for Exploiting the 70-ft Series

The structure and physical qualities of the oil shale indicate that the 70-ft series can be safely mined in open stopes; studies indicate that this method would be the most economical one to use.

Many variations exist in the application of the open-stope principle of mining. The manner of laying out an open-stope mine will depend largely upon the haulage system and other mining practices to be followed. The mining practices, in turn, mostly will be governed by the maximum spans that the back will stand without spalling or caving. The size of pillars will, of course, be governed by the requirements for supporting the roof. The pillar pattern, however, in part may be arranged for convenience of operation.

For an oil-shale mine to be commercial, at least in the immediate future, unusually low mining costs will be necessary. To attain this end a very large output per man-shift will be required, as wages will be the major item of expense. Present plans call for using units of equipment as large as underground conditions permit. Low retorting costs, of course, also will be necessary in a commercial installation.

In general, the unit costs of mining in open-cuts are less than in underground mines. Open-cut or quarry costs must be approached to make the exploitation of the oil shale commercial. The aim, therefore, has been to take a quarry underground and adapt as many surface mining practices as practicable to underground conditions.

Pencil studies indicate that the oil shale can be produced most economically by the room-and-pillar variation of the open-stope method of mining. The second choice of the manner of mining the shale would be in rooms with rib pillars between. A conveyor system for transporting the broken oil shale probably would best fit a mine laid out with rib pillars. Trucks would be more efficient in room-and-pillar workings.

Systems of mining with haulage drifts under the beds to be mined have been considered. The cost of the development work, however, would be about the same as the direct cost of mining the shale where all workings are confined to the beds being mined. In a commercial operation, starting on the outcrop, very little work coming under the classification "development" would be required. Shafts would, of course, be necessary for underground mines back on the mesa.

Consideration has been given to a mine lay-out by which the oil shale would be blasted by means of long drill holes. The practice has many operational advantages and one serious disadvantage, namely, a considerably higher cost per foot of drilling as compared with conventional rock drills. A diamond drill has been purchased and is in use at Rifle. Present indications, however, are that we will not be able to get a low enough drilling cost to make long hole drilling practicable.

Width of Rooms

The first factor to determine in laying out an open-stope system of mining is the maximum width that rooms would safely stand without spalling The oil-shale beds are remarkably homogeneous in horizontal extent, with only rare vertical planes of weakness. Usually a joining plane dies out in a short vertical distance or does not pass from bed to bed. Barodynamic tests of

such a rock should give reliable results regarding safe spans. An investigation, on Bureau of Mines account, of the physical qualities of the oil shale by Prof. Philip B. Bucky and Frederick D. Wright* is nearing completion at Columbia University. A separate paper will be published giving their results. In general, their tests show that the roof stone should stand safely over a span of 50 ft where minimum thickness of individual roof-stone bed is 3 ft.

A test room 8 ft high, 50 ft wide, and 100 ft long (Fig 3) has been run at the oil-shale mine immediately under the roof stone. The room has stood without movement or spalling since mid-December 1946. We feel confident that the roof stone will stand safely over a minimum of 50-ft spans; we do not know yet, however, how long it will remain safe when exposed to ventilating air currents. The rooms that will be used as haulageways may be driven at less than standard widths. Two instruments, using the geophone principle, have been set up to record any rock stresses that might develop in the test room. The technique and instruments were developed by Leonard Obert. Later the room will be widened progressively to 100 ft. We expect to obtain data of the ultimate width that a room may be worked safely and at what point stresses are set up which might cause spalling. The rooms when completed will be 70 ft high. The back of such a room is admittedly high; rooms up to a height of 90 ft, however, are safely carried in underground limestone mines. A mine will be laid out in such a manner that sections could be completed and then abandoned before time could affect the roof stone.

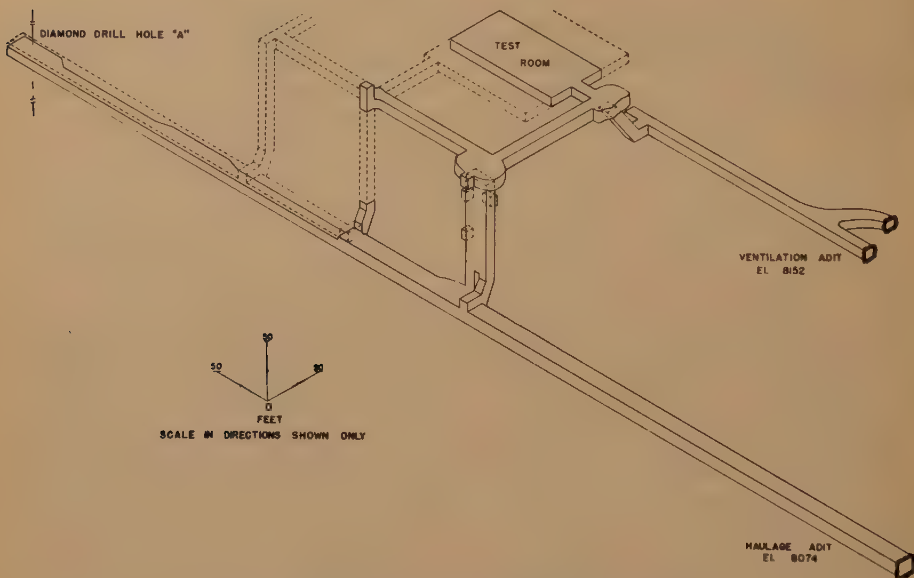

Fig 3—Selective mine area showing progress.

Size of Pillars

Bucky and Wright are now working on the pillar problem. After the theoretical size of the pillar required to support the overlying formation is determined, a pillar arrangement will be decided upon.

* This volume, p. 352

Benches

Two 18- by 26-ft entries have been extended about 100 ft in the oil-shale bed

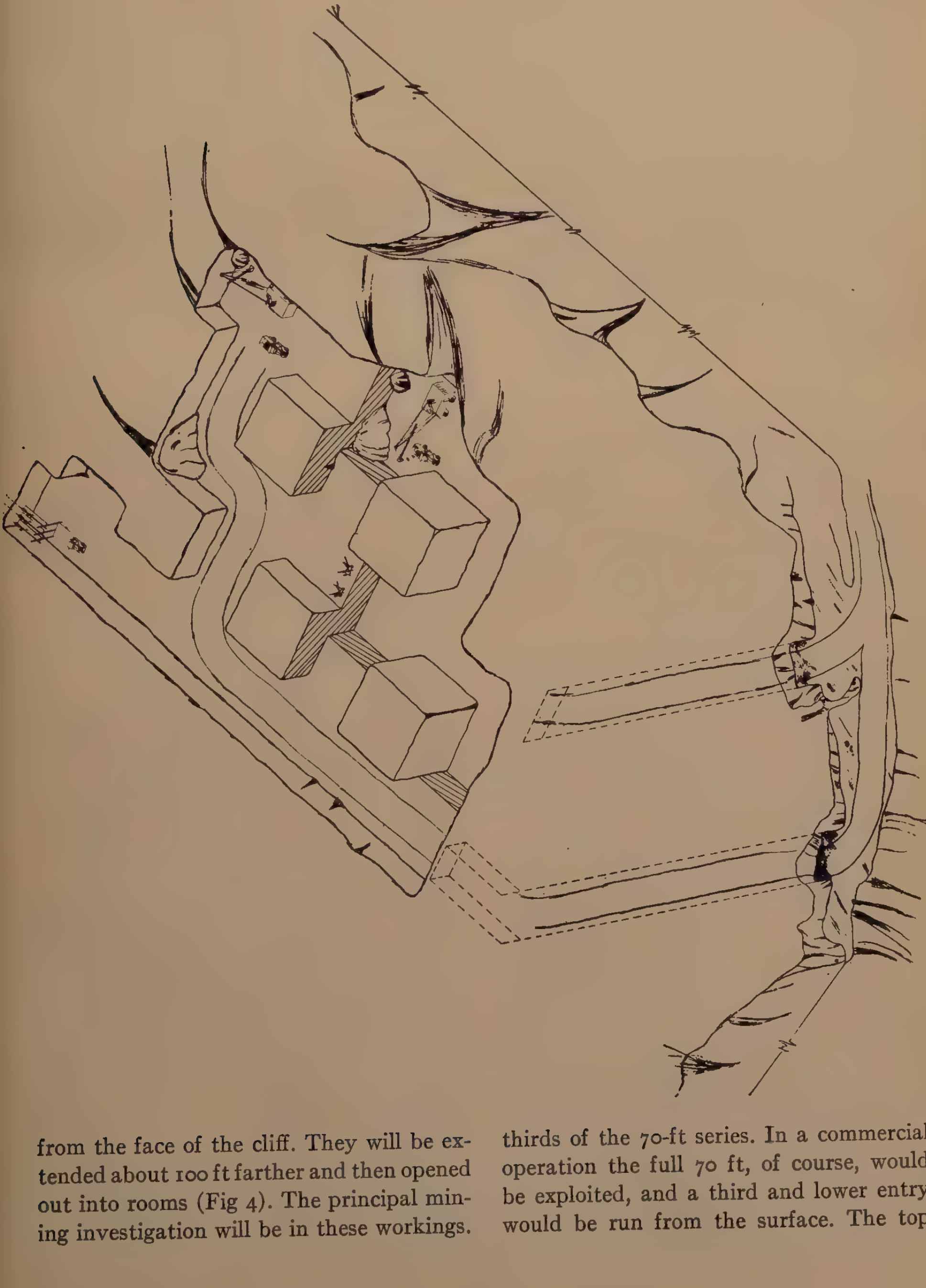

FIG 4—AIR-VIEW DRAWING OF PROPOSED ROOM-AND-PILLAR WORKINGS AT MINE, RIFLE, COLORADO.

from the face of the cliff. They will be extended about 100 ft farther and then opened out into rooms (Fig 4). The principal mining investigation will be in these workings.

As shown by Fig 4, it is planned to develop only two benches in the experimental mine and confine the work to the upper two-thirds of the 70-ft series. In a commercial operation the full 70 ft, of course, would be exploited, and a third and lower entry would be run from the surface. The top

heading in the experimental mine will be 25 ft high and the other bench 22½ ft high; the third bench, if run, would be 22½ ft high. Experimental data and cost figures obtained on the middle bench should apply equally well to the lower one. The blast holes in the top bench will be drilled horizontally; vertical, downward, parallel holes probably will be used in the lower benches. All drilling in this phase of the experimental work will be done with air drills. The breaking costs in the advance heading will be higher than in the succeeding benches because rounds will have only one free face to which to break, whereas rounds on benches would have two free faces.

Loading

Where the necessary working space is available, the most efficient loading unit is an electric shovel. This type of loader (electric or Diesel drive) is used almost universally in open-pit metal mines and hard rock quarries; electric shovels also are commonly used in underground rock mines. The ability and efficiency of a shovel to handle coarse material increase with the size of the unit; moreover, the tons per manshift handled increase with the size of the shovel. It was developed at conferences with shovel manufacturers that a standard 2½-yd shovel was the largest practical unit that could be worked in the shale mine. As the oil shale is of a relatively low specific gravity, a 3-yd dipper can be used on the 2½-yd model. As the digging is expected to be tough, a Ward-Leonard control on the shovel was considered desirable and necessary. A shovel of the above specifications, with a special front end that will permit working under a 25-ft roof, has been ordered. The 25-ft height of the top bench is a compromise. The standard model requires 27½ ft of headroom, and the most desirable height from the mining standpoint would be 23⅓ ft (one-third of 70). A special front end could be designed for the shovel to work under a 23⅓-ft height, but the shovel efficiency, particularly in handling coarse material, apparently would be decreased to an extent that would more than offset the gain that would result in working a 23⅓-ft advance heading instead of one 25 ft high.

Underground Haulage

Three standard methods of haulage are used in underground rock mines: (1) cars running on tracks, (2) trucks, and (3) belt conveyors. Mines are laid out to fit the haulage system, and haulage practices must fit the manner of loading. Broken rock can be moved on conveyors at the lowest cost per ton-mile, but there is a limit to the size of material that can be handled economically on belts, and the oil shale undoubtedly will break into coarse sizes. Moreover, a power shovel cannot load material as blasted down directly onto a main line conveyor; an intermediate step would be necessary. Considerable study was given to belt haulage for the shale mine, but the idea was abandoned reluctantly. Crushing the rock before loading it on the conveyor would not be practicable.

Where large tonnages can be loaded at a single point, a lower cost per ton-mile can be obtained by rail haulage than with trucks; rail haulage, however, does not lend itself to room-and-pillar mining. Shovels commonly load freshly blasted material in open-pit mines into railroad cars. In such cases the track runs parallel to the bench and a train of cars can be loaded one after another without switching. In underground workings, however, stub tracks are necessary to get the cars to the face; and only one, or at the most two, cars can be loaded without switching. The cost of switching and loss of capacity of a shovel while waiting for cars appear to offset any advantage of costs that would be gained with rail haulage instead of trucks. A decision has been made to use Diesel trucks for hauling the oil shale;

two 15-ton Diesels are on hand. Full consideration has been given to possible fouling of the mine atmosphere by the exhaust fumes. An adequate ventilation system is planned for the mine workings; moreover, it is considered that the exhaust gases would be adequately diluted in the large rooms.

Drilling and Blasting

The daily capacity of the shovel used for loading the broken oil shale and the size of the dipper will influence the type and depth of stope rounds. A round should provide enough broken stone for a full shovel shift; the stone should be broken down in primary blasting to a size that will pass through the shovel dipper. The capacity of the shovel is tentatively being taken as 1200 tons per shift; the inside dimensions of the dipper are 40 by 59 in.

No actual experimental work has yet been done on the type and depth of stope rounds because of the lack of working places. Moreover, for the same reason, no experimental work has been done with explosives to be used in the stope rounds. A jumbo with wagon-drill mountings is being developed to hold a battery of drills, for horizontal drilling. Wagon drills probably will be used on the lower benches.

Drilling apparently will be the largest individual item of expense. Work to date in development headings indicates that about 24 in. per minute is drilled with 3½-in. piston machines with an air pressure of 95 psi in holes up to 6 ft deep. The drilling rate, however, decreases to 14 in. per min. in holes 18 ft deep. This latter rate is not satisfactory; the rate in the first part of the hole could be considered satisfactory, but an increase would be desirable.

It has been found that commercial detachable bits are dulled principally by losing gauge and have to be changed after about 5 ft of drilling. As deep holes will be used in all rounds, the drilling cost would be too high for our purposes if bit changes were required every 5 ft. Moreover, the rapid loss of gauge would tend to cause drill rods to stick in the long holes. Plans call for drilling 22½-ft holes with no more than one change of drill rods and with only one man on a machine. A bit has been developed at Rifle that will drill over 100 ft without serious loss of gauge. The use of this bit has reduced the drilling cost; although there is a reduction of cost of bit per foot of hole, the great saving is in labor.

In an effort to lower drilling costs, a fundamental study is being made of the whole drilling problem. Apparently there is a speed of rotation of the drill that will give optimum results in the oil shale; this is being investigated. Work is also progressing on the shape and form of bits to increase drilling speed. As other investigators have found, grinding of drill cuttings in a hole reduces the drilling speed. Increasing the water supply in a standard drill increased the drilling speed, doubtless by flushing the drill cuttings away faster. Our aim is to develop a bit that will make a minimum of fines in cutting the shale. Drilling speed depends directly upon the air pressure; with too high a pressure, however, drill and bit breakage offsets the advantage of the faster rate. Some curves have been drawn, but the final answer has not yet been obtained; the range apparently is 95 to 105 psi at the drill. The decrease in drilling speed with depth is a serious factor. Present studies are mostly in the theoretical stage. Alloy steel tubes are being tried as drill rods; these lighter rods should require less energy to overcome their inertia.

Summary

The principal job of the Oil Shale Mining Division is to select methods and devise practices for producing the oil shale on a commercial scale at the lowest practicable cost. The preliminary thinking on the problem largely has been completed; the method for mining the shale has been decided upon; and the loading device

and the haulage equipment have been selected. Considerable major research remains to be done on drilling and blasting problems, on integrating breaking of the shale with loading and with haulage, and in establishing costs. Mining research to date has been determining the safe span that the oil shale will stand in open rooms and on drilling problems. The work is well organized, and procedures for finishing the job have been outlined.

General

The petroleum industry in 1946 made a new production record in the United States of 1,731 million barrels, an increase of 1.2 pct over 1945. This output exceeded the domestic demand. As long as this condition exists there will, of course, be no need for synthetic liquid fuels. Our known petroleum reserves, however, are limited and are not being increased notwithstanding the most extensive drilling campaign in the history of the industry. Unless new major fields are discovered or the country goes into a long depression, it is my opinion that an oil-shale industry will be established in the next decade.

We expect to demonstrate that oil shale can be mined and retorted commercially, although not perhaps while petroleum sells at the present price. A moderate increase in the price of petroleum, however, should bring shale oil into the picture.

The thought, of course, arises that when the time comes when domestic production of petroleum does not equal domestic demands the deficiency could be made up by imports. From the standpoint of national defense, however, an adequate dependable supply of liquid fuels within the boundary of our own country is imperative. You and I could run our cars with gasoline from Iran without an uneasy thought, but our thoughts would be far from easy if our Air Force or Navy had to depend upon this source in case of another war.

We do not think of shale oil as competitive to petroleum, but rather as supplemental. The cost of finding petroleum is increasing, and when this added cost is reflected in the price of the marketed products the field will be open for shale oil—an unimportant amount at the start but one that could be increased to meet the demand. A shale-oil industry has not yet been born, but it has been conceived. Perhaps it will have to be nursed in infancy but should grow stronger as efficiency is gained by experience.

As stated, an underground mining unit would be about 20,000 tons per day to produce 10,000 bbl of oil. The daily production of petroleum is nearly four and three fourths million barrels per day. Close to 500 underground oil shale mines, each with a capacity of 20,000 tons per day, would be required to produce this amount of oil; or, if open-pit mining was followed, the daily capacity of each mine would roughly be 40,000 tons. The oil-shale reserves in Colorado would last well over 100 years at this rate.

One of the difficulties of establishing an oil-shale industry is that it lies in two unrelated fields. Retorting, refining, and marketing of the products that would be obtained are beyond the experience of most mining concerns, and most oil companies are unfamiliar with mining problems; it is hoped that the work now under way at Rifle will bridge this chasm.

Recent Developments in Mechanization at the Bunker Hill Mine

By R. S. Hooper*

(New York Meeting, March 1947)

In attempting to describe recent mining developments at the Bunker Hill mine, it may be well to recall first the old days of hand mining when holes were laboriously drilled by hand to a maximum depth of three feet, and were shot one or two at a time, in order that the miner could take advantage of every slip, crack or bedding plane, and so save as much drilling as possible.

Broken muck, of course, was loaded by hand shoveling into wheelbarrows or small cars, and heading advances or stope tonnages were governed by the number of men that could find room to work in any given place.

The first big improvement over these old methods was the introduction of the reciprocating type of air-powered rock drill. This innovation, crude as it was, enabled the miners to set up a machine and drill a complete round of holes in one shift; but broken muck was still loaded by hand, and the size and rate of advance in headings were limited by time required to remove the broken rock.

The next improvement of consequence was the appearance of the hammer-type rock drill, which greatly increased drilling speed over the reciprocating type, and made possible the completion of a full cycle in one shift, by the use of crossbars in headings, and permitted drilling and mucking to be carried on at the same time.

Rock drills continued to improve as time passed, and finally the smaller and more efficient mucking machines appeared on the scene, and it became possible to muck out a round of broken rock and have a clean setup for one or more drill columns and machines. A complete cycle of mucking, drilling, and blasting was completed in one shift with an average advance of five feet.

These conditions prevailed for several years, but as the mine grew deeper, the distances to transport the workers grew longer, and this fact together with the shortening of the working day, soon presented the problem of attempting to complete a cycle of operations in a shift of only six hours at the working faces.

Anticipating this condition, the operating heads at the Bunker Hill had started an intensive study of more and better mechanization, and over a period of comparatively few years have adopted and placed in service methods and machines that make it possible for a three-man crew to consistently advance headings seven by nine feet in cross section, at the rate of eight feet, and more, per shift, doing all the mucking, switching, drilling and loading.

These improvements consist of the following major items:

1. The development of a light, flexible, easily transported jumbo equipped with pneumatic columns (Fig 1), that has decreased the "setting up" and "tearing down" time an average of 1½ hr per shift, or 25 pct of the available working time. These jumbos were developed at the Bunker Hill mine by members of the staff and consist of two pneumatic columns attached to a suitable carriage. They fold

Manuscript received at the office of the Institute Jan. 9, 1947; revised May 19, 1947. Issued as TP 2256 in Mining Technology, September 1947.

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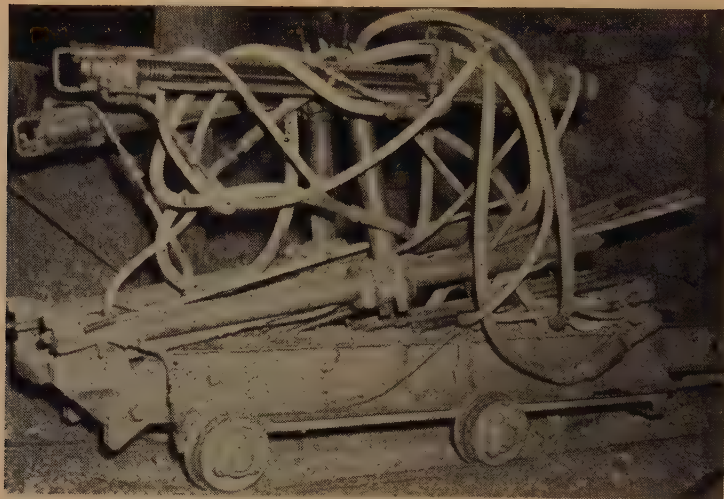

FIG 1—TRACK-MOUNTED JUMBO. FOLDED FOR MOVING.

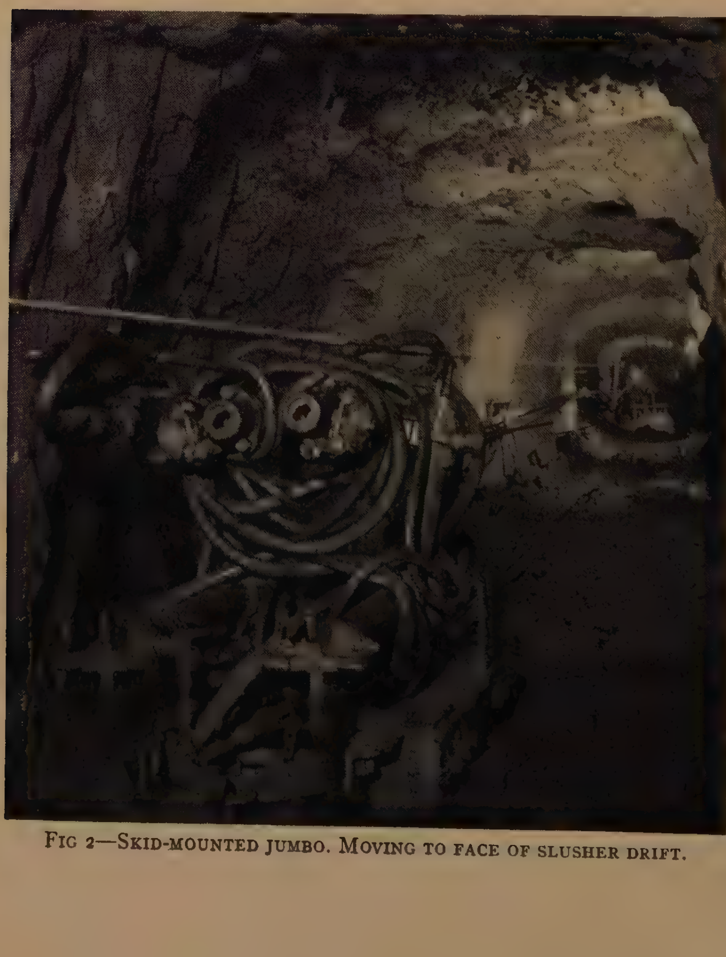

FIG 2—SKID-MOUNTED JUMBO. MOVING TO FACE OF SLUSHER DRIFT.

up easily for transportation (Fig 2) and when set up they require no roof jacks or track clamps as all the weight of the machines is carried on the columns (Fig 3). The total weight of our jumbos including two DA-35 automatic feed drifters is 2200 lb, and a safety feature is the Ingersoll Rand check valve installed on each column which prevents collapse of the column in case of air failure. These jumbos have been patented and are now manufactured and sold by Ingersoll Rand Company.

2. Installation of slushing systems that practically eliminate hand mucking, and slusher hoists are mounted on turn tables that can be set at any angle through the full 360°.

3. Larger and faster mucking machines. The mucking machines used are Gardner Denver model GD 9 and Eimco 12 B and 21. The model 21 is now used in headings where rapid advance is desired as the mucking cycle is definitely shortened by the use of the larger machine.

4. Larger, heavier locomotives with greater storage battery capacity. Our locomotives for heading work are 6-ton machines, and are normally equipped with a battery of 72-A8 Edison cells, but we are now equipping some batteries with 84-C7 Edison cells with a consequent increase in power and capacity.

5. Simple, efficient "cherry pickers" that can be installed with a minimum of time and labor. These "cherry pickers" were also developed at the Bunker Hill mine and consist of an *I* beam of suitable section suspended from the back of the drift, and this *I* beam is equipped with a crawl and an air cylinder. The cars are raised from the rails by the air cylinder and are pushed to one side to clear the tracks and require only room enough for one car to pass the rest of the train.

6. Greater car capacity, so switching and tramming time is reduced. Cars range from 2½ to 5 tons capacity and are of the bottom-dump type. They are built and repaired in our own shops.

7. Improved trackage with heavier steel, to handle heavier loads and greater speeds. Steel is from 30 to 85 lb section. The lighter sizes being used in crooked drifts and on sub levels and the heavier sections for main-line haulage.

Fig 3—Track-mounted jumbo. Drilling position.

8. Air lines of ample capacity to maintain good working pressure. Eighty to eighty-five pounds working pressure at the machines is considered standard.

9. Automatic feed machines and detachable bits. Several makes of machines are used in the mine but all jumbos are equipped with Ingersoll Rand DA-35 power feed drifters. Detachable bits are Ingersoll Rand "Jack Bits" and are

reconditioned in our shops by hot milling and a salt draw.

10. Improved ventilation of all headings, with spot coolers provided in the warmer places, and the cooled dehumidified air conducted to the faces through metal ducts.

11. Improved cycle of operations.

A time study of a typical crosscut, seven by nine feet in cross section, driven through medium-hard quartzite is shown in Table 1.

Two-man crews working under a suitable incentive system have been tried, and capable men have advanced headings at a rate of over 6 ft per shift in rock, and 5 ft in timbered drifts. They also have completed the cycles in the 6-hr working time at the faces. These two-man crews also do all mucking, switching, drilling and loading.

In slusher and scram drifts where no trackage is used, the pneumatic column jumbos are skid mounted (Fig 4), and are

TABLE 1—*Time Study No. 22 Level—Three-man Drift Crew*

Leave portal.......	7:15 a.m.	
Arrive level........	7:52 a.m.	
At work...........	7:53 a.m.	Take train from station
Start mucking.....	8:01 a.m.	
Mucking finished..	9:30 a.m.	Dumped last cars: got steel, bits and jumbo
Arrive face........	9:38 a.m.	Connect hoses; set up jumbo
Start drilling (2 machines)	9:54 a.m.	One man goes for powder during drilling time
Finish drilling.....	11:36 a.m.	22 holes for 7-ft round
Tear down and remove jumbo.....	11:41 a.m.	
Start loading......	11:45 a.m.	Using electric delays
Finish loading.....	12:04 p.m.	

Note: Drift crew left, and track men start advancing permanent track. Track men will throw blasting switch upon completing track advance. Drift crews are allowed to leave the mine upon completion of the cycle.

FIG 4—SKID-MOUNTED JUMBO. DRILLING POSITION.

transported to and from the face by the same slusher hoists that are used to remove the broken rock; and in some instances three-man crews have advanced these five by seven foot headings at the rate of over ten feet per shift—doing all drilling and slushing.

In shaft-sinking operations up to 50° the pneumatic columns have also been adapted to this type of work; power-feed drifter drills are mounted on the columns and lugged steel and detachable bits are used as in drift and crosscut advance.

Raises up to 41° are also driven with this type of equipment and in the mine, stoper-type drills are seldom seen in action.

Stoping operations in the Bunker Hill are carried on by the square set and fill method, and here again the pneumatic columns are in general use. Single columns are used and provide a safe, steady, and quick setup for either 3 or 3½ in. drifter drills of both the hand-cranked and power-feed types. Heretofore, the saddles have been mounted directly on the pneumatic columns and the drills operated on their sides. At first thought this method might suggest that there would be excessive wear on the drill guides and shells, but careful checking has proved that such is not the case. Single pneumatic columns equipped with standard arms and used with drifter drills, are going into service in the stopes (Fig 5). They give every promise of being entirely successful. These single stope columns are not only used with saddle mounting, but also equipped with the safety check valve

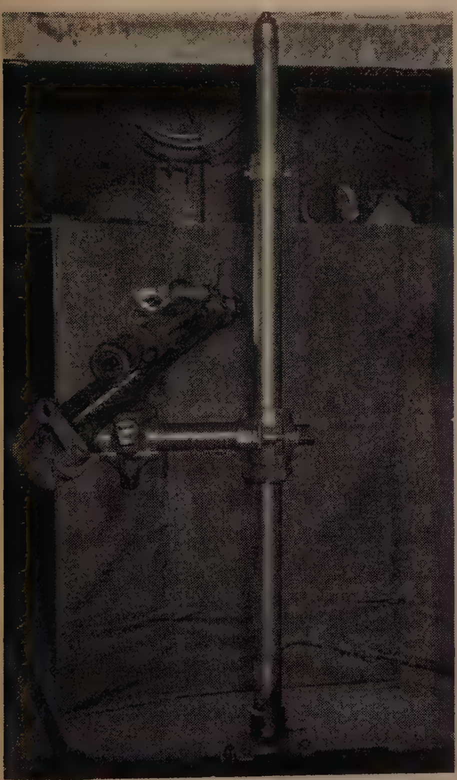

Fig 5—Single column with arm mounting.

Fig 6—Single-column machine assembly.

previously mentioned, and when correctly set up resist twisting fully as well as a screw-type column (Fig 6). They never loosen as air pressure keeps them tight at all times. The 6-foot model weighs 91 lb as against a weight of 125 lb for the standard 6-foot screw-type column.

So while many types of machines and equipment have been adopted and put to work with the idea of making the best possible use of the reduced working time in the various places, one of the greatest savings of time has been accomplished in the many and varied adaptations of the pneumatic column.

Acknowledgments

I wish to thank Mr. J. B. Haffner, General Manager, and Mr. S. W. McDougall, Mine Superintendent, respectively, of the Bunker Hill and Sullivan Mining and Concentration Co., for general help and constructive criticism in the preparation of this paper.

Use of Steel in Top Slicing

By James L. Bruce,* Member, George W. Nicolson,† and John G. Tate,‡ Member AIME

(Los Angeles Meeting, October 1947 and New York Meeting, February 1948)

For more than 25 years modern mining has been carried on in the Island of Cyprus, Mediterranean Sea, by the Cyprus Mines Corp. of Los Angeles, Calif. The general features of these operations have been described.[1]

From 1935 to June 30, 1940, top slicing in the Mavrovouni mine produced in excess of 2,000,000 long tons. The rate for the last part of this period was approximately 50,000 tons per month.

Description of Ore Bodies

The shape of the ore body now being top sliced in the Mavrovouni mine is an inclined irregular pyritic pipe or chimney of large dimensions, extending from unknown depths below the bottom level at elevation 250 ft below sea level, to the highest level at elevation 400 ft above sea level. The present surface above the ore body lies between 550 and 650 ft elevation. Undoubtedly the ore body extended to surface and was mined by the ancients by opencast and underground methods. The old pits were gradually filled up by erosion until now they show as slight depressions in the topography. The largest area of the ore body is at elevation 170 ft below sea level, where the horizontal section is almost elliptical, with a width of 600 ft and a length of 1000 ft. Below this elevation the shape of the ore body continues to be elliptical but with reduced dimensions. Above this elevation it rises flatly towards the south with a substantial depression in the upperside of the chimney and irregular ridges or wings rising on each side. It gradually grows smaller as it rises.

The ore body is a pyritic replacement of andesitic extrusives which occur as flows and pillow lavas. With the exception of the upper portions of one or two small offshoots rising from the main ore body, all ore is completely surrounded by andesite. Most of the andesite lying above the ore body is covered by marl and clay sedimentaries of Miocene and later age. The Miocene is presumed to be older than the ore mineralization. That portion of the andesites which lies between the sedimentaries and the ore body has been broken by slumpage, and weathered, and iron-stained by the descending meteoric waters. It is a mass of crushed material that caves and follows downward readily as the top-slice stopes retreat.

The main ore body is a heavy cupreous pyrites mass assaying on the average about 4.0 pct Cu; 48.0, S; and 42.0, Fe. In places the copper drops below 1.0 pct. On the borders of this heavy sulphide ore body lie disseminated cupreous pyrites ore deposits of large dimensions. Some portions of these appear to be closely related to the heavy sulphide ore, as the ratio of copper to pyrite is about the same. The greater part of the disseminated ore, most of which lies immediately to the east of the main ore body, contains less than 0.10 pct Cu and about 25.0 to 30.0 pct S. This probably can

Manuscript received at the office of the Institute Oct. 24, 1947. Issued as TP 2458 in Mining Technology, September 1948.

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[1] References are at the end of the paper.

be mined cheaply by caving methods, and concentrated cheaply by gravity and flotation to give a low-copper, high-sulphur pyrite concentrate.

The heavy sulphide ore is not the dense homogeneous pyrite commonly found in pyrite mines, which occupies 8 cu ft or less per ton, but is a loose crumbly porous mixture of coarse and fine pyrite and occupies 10 to 10.5 cu ft per long ton.

Development

During the early period of prospecting, the ore body and adjacent area was thoroughly churn-drilled by 50 drill holes. The locations of the hoisting shafts, ventilation shafts, and main haulage levels were determined by study of a model constructed in accord with information gained from the churn-drilling records.

The mine is served by 7 shafts and one adit tunnel driven eastward for a distance of about 3000 ft from the portal at 314 ft elevation above sea level to reach the upper portions of the south end of the ore body. The portal of the tunnel is near the river bed of the Xeropotamos River. No. 2 shaft was sunk from surface to expedite driving of the 314 tunnel and is used now only for entry of air. No. 3 shaft, vertical, with collar-elevation of 615 ft, is used as a main shaft to hoist ore from stations at *A* level, 125 level, and 200 level, to the 314-tunnel level. Levels situated below sea level are named with the letters of the alphabet. *A* level is 20 ft below sea level. Men are hoisted and lowered between surface and these levels. No. 3 shaft intersects 314-tunnel level 1700 ft from its portal and about 600 ft west of the west side of the ore body. It is possible that this shaft will eventually be damaged by surface subsidence when mining the lower parts of the ore body, consequently No. 6 shaft, which is another vertical main hoisting shaft, with collar at 310-ft elevation, has been sunk from point near portal of 314 tunnel to *D* level at minus 170 elevation, which is the mine's principal tramming level.

No. 7 shaft with collar at 307-ft elevation is sunk on 30° incline to *D* level. It is partly unlined, partly timbered, with clearance of 9 by 9 ft. It is used principally as an air shaft and is provided with tracks and a stairway to give ready exit or entry for men and supplies. *D* level, to which it connects near the No. 6 shaft station, has two parallel galleries connecting to the west side of the ore body. These are circular, smooth, concrete-lined galleries 10 ft 6 in. id. Shafts Nos. 3 and 6 are cylindrical, concrete-lined, 14 ft 9 in. diam, with steel ladders and supports and no wood except the shaft cage guides. No. 4 shaft, vertical, 14 ft 9 in.-diam, cylindrical, concrete-lined, without ladders, with collar at 692-ft elevation and bottom at *A* level, is southwest of the ore body. It is used only as a foul air exit shaft. No. 5 shaft, vertical, with collar at 628-ft elevation and bottom at 200-ft level is south of the ore body. It is 10 ft 6 in. in diameter, cylindrical, concrete-lined, with steel ladders.

Paralleling No. 3 shaft and extending from 314-ft level to *D* level is No. 1 winze which is equipped with hoist for men and supplies. This is cylindrical, concrete-lined, 10 ft 6-in. diam.

Most of the galleries in country rock are smooth bore, cylindrical, concrete block lined, 8 ft in diam to 10 ft 6 in. diam. A typical section is illustrated elsewhere.[2]

Inside the ore body the mine is served by several 30° incline shafts, connecting from highest levels to lowest levels in such manner that men from any part of the mine can walk out to the surface in case of emergency. Some of these inclines are equipped with tracks, skips, and cages, for handling ore, men, and supplies. Main haulage levels are at elevations 314 ft, 200, 125, minus 20 (*A* level), minus 170 (*D* level). Lowest developed level is at minus 250 ft. Between these there are many sublevels, some of

which are used exclusively as foul air returns.

Many levels in the mine are provided with fire refuge stations equipped with doors, air and water lines.

Ore chutes are usually systematically spaced at intervals of 35 to 50 ft in both directions. Most of these are constructed as vertical, concrete block-lined cylinders. They are 8 ft 6 in. diam for 15 ft above the bottom. Above this they are reduced to 4 ft 8 in. The 4 ft 8 in. section sometimes is lined with hardwood blocks to give 30 in. id, in order to take the wear of falling ore. The wear is taken on the end grains and the space between the wood blocks and the concrete blocks is securely packed with wet clay. Chute lips are of steel, bolted for easy removal, and fitted with arc gate (Fig 1).

Transport of Ore

Track is 30-in. gauge with heavy rail. Ore cars, of special design for rotary tipple dumping, hold 4.5 long tons of ore. Waste is handled in rocker dump cars of three long tons capacity. All cars are equipped with Timken bearings and MCB ½ size automatic couplers. Cars are hauled in trains of 10 to 25 cars by Westinghouse 6-ton electric storage battery locomotives equipped with Edison batteries.

Cars are caged and uncaged semi-automatically by gravity at each principal shaft station and at surface. Tracks on stations close to shaft and on cages are set on grade at about 1½ pct so that cars move forward without manual help when dogs are released by landing cage on shaft chairs. Track grades on stations (not immediately adjacent to shaft) are about 0.4 to 0.5 pct. Dogs are equipped with springs to cushion against shock. Shaft cages are double decked with upper deck for cars, men and supplies, and lower decks for men and supplies only. Each deck accommodates 25 men. Details are shown elsewhere[3] of this and also of the free box-type 5 ton (4.5 long tons) mine car.[4]

Hoisting at No. 3 and No. 6 shafts is by alternating-current, electric-powered, Nordberg double-drum geared hoists. Speed of hoisting is kept low to avoid high peak

Fig 1—Standard steel chute gate.

loads. It can be readily increased by change of motor and gears.

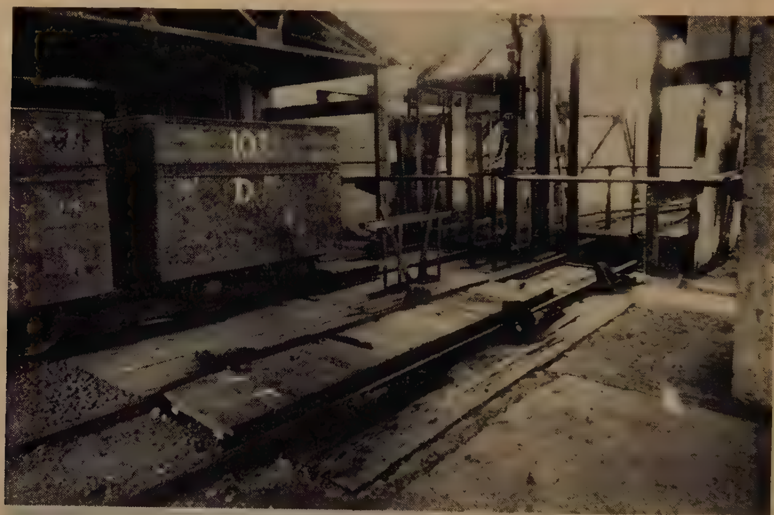
Underground shaft stations are 16 ft wide, concrete, arched, double-tracked, with accommodation for 20 cars on each track on the cage-loading side. On the opposite side empty cars pass by gravity through spring switch, and are kicked back by gravity to a low point in the track where they are picked up by the storage battery locomotives for return to the mine.

On surface, loaded car leaves cage by gravity when cage is set on shaft chairs. At same time empty car runs on cage by gravity (Fig 2). All surface yard handling

of cars, from shaft or tunnel to platform scales, to rotary tipple, and back to shaft or tunnel, is accomplished by gravity on tracks with carefully designed curves and grades. Car retarders are used to prevent over-speeding. Car stops are used at scales and at tipple. After passing tipple which dumps directly into railway cars, mine cars run to low point of yard from which they are elevated to high point by chain haul on 15° incline (Fig 3). From this point cars must be properly distributed to the three different places from which they came. This is accomplished automatically by lugs at three different places on underside of cars. These lugs strike track-switch controls which throw the proper switch.

After passing scales each class of ore is segregated by diversion to siding until seven cars of such class have been accumu-

2



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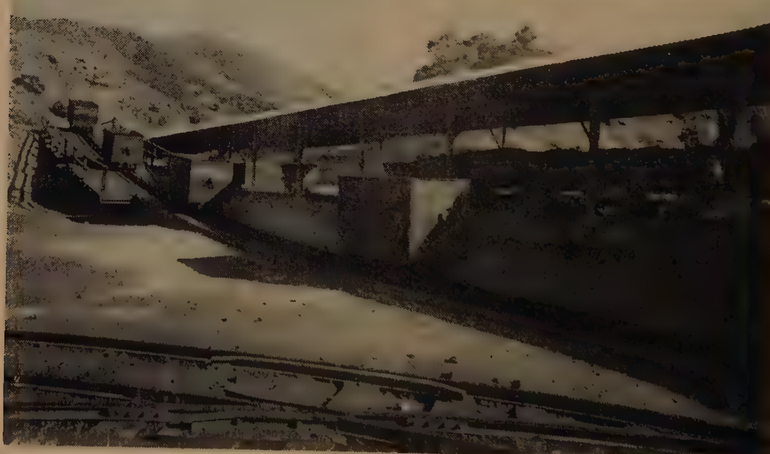

Fig 2—Car caging equipment at collar of No. 6 shaft. On near track is shown close-up view of caging dogs.

Fig 3—Chain car-haul automatically takes cars from point below the tipple and elevates them to high point on No. 6 shaft yards.

lated. This then goes to tipple, to fill one railway car which is marked accordingly. (For tipple design see ref 5.)

This method of hoisting with 4.5 ton cars (long tons) gives nearly the same rate of hoisting as with skips of the same capacity. It has several advantages: it avoids four dumpings of the ore, undesirable because dumping causes oxidation; it is a very much more convenient arrangement for selective mining whenever it is advantageous to produce several different classes or types of ore, as it keeps these classes separated from mine chutes to railway cars; it facilitates separate hoisting of waste from mine to surface; it requires more mine cars but it avoids construction of expensive skip pockets and ore bins.

Each car of ore or waste is identified by a card placed in the small pocket at the end of the car when loading. This shows date, chute number or heading number, class or grade of ore or waste, and sequence number at that loading point. This card is removed and attached to weight card at scale house.

Ventilation

Early recognition of the great importance of good ventilation was a controlling factor in the decision to make most shafts and galleries of circular cross section, and smooth lined with poured concrete or precast concrete blocks. This has made it possible to pass 130,000 cu ft of ventilating air per minute with only 1.3 in. water gauge and very low power consumption. Normally all openings to the mine except No. 4 shaft are used as fresh air inlet shafts. No. 4 shaft is the foul air outlet shaft. Two aerovane-type fans with propeller blades spanning 8-ft diam are used to exhaust foul air through numerous galleries leading to fans from the stope faces. These fans blow the air out through No. 4 shaft. Fans are arranged with two-speed, *V* belt drive and operate, whenever mine is inactive, at the slower speed with considerable power saving. Fan drive is with small motor for slow speed and larger motor for higher speed.[6]

No. 7 incline shaft may be changed from a fresh-air inlet to a foul-air outlet by means of two underground ventilated doors.

Whenever developments or stoping are being advanced into new mine areas, the practice is to advance air inlet and air return together so as to always maintain good air circulation. Main air-return galleries connecting with suction side of fans are usually two or three floors (top-slice floors) above tramming levels.

Booster fans of the aerovane type are installed in the air returns on suction side of main fans in such sections of the ventilation system as require more volume of air than can be obtained with the relatively low-water gauge of main fans. This keeps total power requirements at the minimum.

Pumping

Mine water is corrosive with acid ferrous ferric cupreous sulphates. Water is raised to surface by triplex bronze pumps through asbestos cement high-pressure pump discharge lines.

Top Slicing with Steel

When the Mavrovouni mine went into production, three mining methods were tried for comparative costs and results. The three methods were cut-and-fill with horizontal cuts, cut-and-fill with vertical cuts, and top slicing with steel posts and rail matting.

After extensive trials of the best practices which had been developed at Skouriotissa and Mavrovouni mines for cut-and-fill stopes, both horizontal and vertical slice types, it was recognized that a satisfactory method of top slicing would have marked advantages, especially in avoiding moving and fracturing of the ore, which occurred to a limited, but undesirable, extent when mining by cut-and-fill methods.

Such fracturing opened up the ore to oxidation and caused heating of the massive sulphides and occasionally fire, especially when sulphides came in contact with timber.

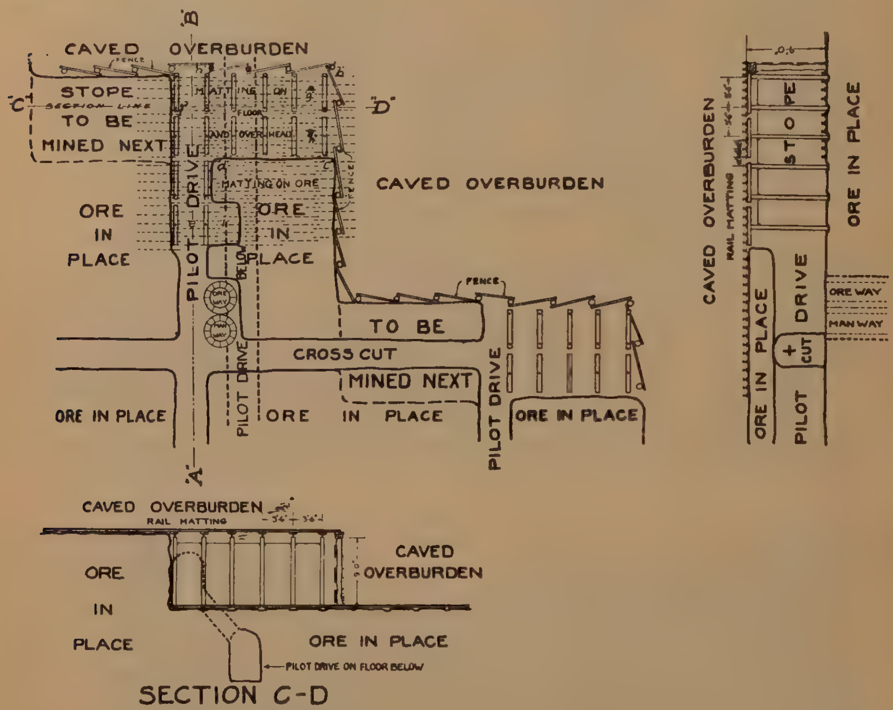

FIG 4—TOP-SLICE MINING, PLAN AND SECTIONS.

This had not become a serious problem in the Mavrovouni mine but it was believed, as a result of experience in the older Skouriotissa operations, that such conditions would grow progressively worse as more stopes were opened up and as filling became more extensive throughout the mine. Also, it was realized that the difficulty and expense of getting filling into the stopes would become greater after the narrow ore bodies of the upper levels were worked out and as the overburden of the lower and wider ore bodies became badly broken up by subsidence. The fact was recognized, however, that the overburden of the top-slice stoping would contain some sulphides and that it would be very risky to develop the customary top-slice stoping method with a thick mat of broken timber between the ore and the overburden. The oxidation of the ore and the sulphides in contact with the timber would almost certainly result in disastrous fires in the timber mat with grave risk of losing all the bodies of ore under the timber-matted area in which such fires might occur.

During early development period, the hoisting shafts, ventilation shafts and galleries, haulage levels, occasional raises and crosscuts, were driven before top slicing was selected as the mining method.

A complete set of maps was on hand showing plans and vertical sections in two directions with all important information from the drill-hole records and considerable underground work. From these maps various groups of top-slice stopes were located, showing the position of chutes

and manways, pilot drives, pillar lines, direction of travel, sequence of mining, and so forth.

A slice thickness of 9 ft was selected as being the highest practical slice suitable for our class of inexperienced native labor.

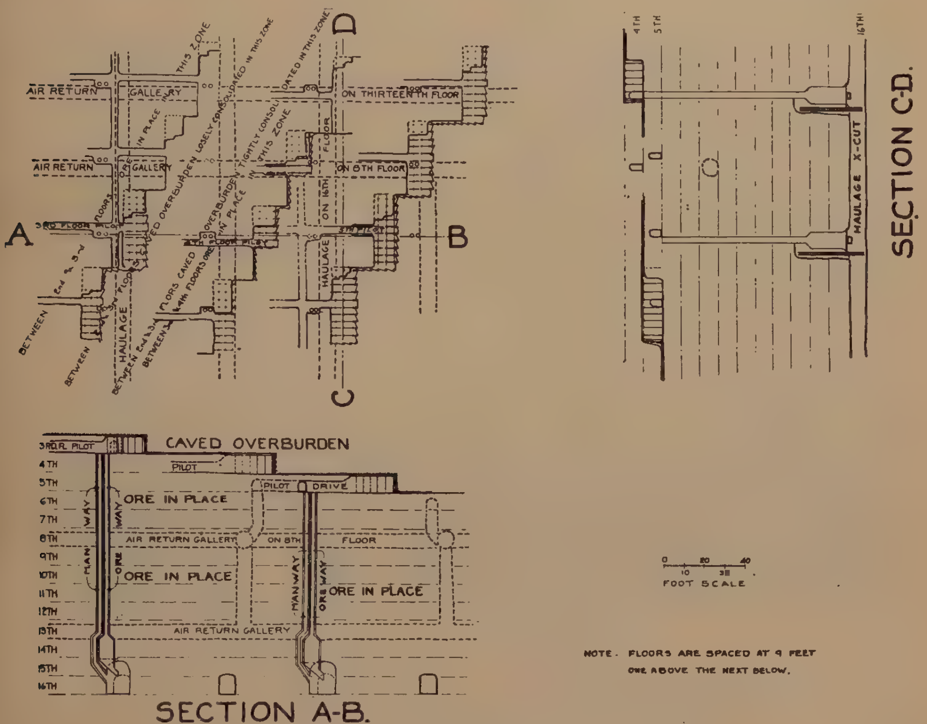

FIG 5—TOP-SLICE MINING, PLAN AND SECTIONS.

From the highest haulage level several vertical two-compartment raises were driven at about 100-ft intervals. These raises were driven to the roof or capping, or even into the capping, in order to be certain that no detached masses of pyrites were left behind before caving the roof.

In the upper part of these raises nails are placed by the engineers to indicate floor elevations, and crosscuts and pilot drives for the highest floor are driven from the manway side. The top floor should have more than 4 ft of ore, and less than 13 ft, upon which to start driving.

Pilot drives are driven parallel to the long axis of the ore body and crosscuts are at right angles with them. Both are driven level, 4 ft wide and 6 ft high. When ground is heavy 3 ft of unbroken ore is left between the roof of the drives and the matting of the floor above. If the drives require timber supports, the dimensions given refer to inside measurements of the timbers (Fig 4 and 5).

The pilots and crosscuts are driven to the limits of the ore body or to a predetermined line. Pilots are driven at 36-ft intervals and crosscuts at 50-ft intervals.

As top slicing proceeds, raises from the haulage levels are driven as needed for chutes and manways.

When the outline of the upper floor has been determined by pilots and crosscuts, and if raises show that no pyrite is left in the roof, then the ore may be mined out in small stopes at right angles

with the pilot drives and parallel to the crosscuts, using wood posts and headboards to support the roof until the ore has been extracted. Steel matting is laid on the floor before allowing the roof to cave.

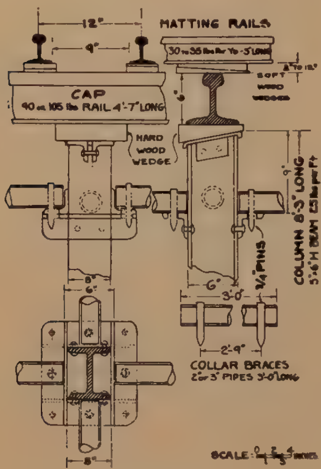

Fig 6

The steel matting which is used in lieu of the customary timber mat is actually a floor *lagging* which is recovered and used again. It in no way resembles the mass of matted timber which is allowed customarily to accumulate in conventional top-slicing operations. The steel lagging has been given the name matting arbitrarily. It consists of 5 ft lengths of 30 to 35-lb rail placed on the floor on their bases and spaced 12 in. apart center to center. This leaves a 9 in. open space between the rail bases. The rails are laid at right angles to the pilot drives. Lengthwise rails are lapped about 6 in. on each end by placing one group of rails in the spaces between ends of rails in the next group. After matting rails are laid on the floor, they are covered with about 6 in. of clay from the cave or roof and sprinkled with water to make a moist mass that will pack and consolidate into a good roof for the next lower slice. One ton (2240 lb) of 35-lb

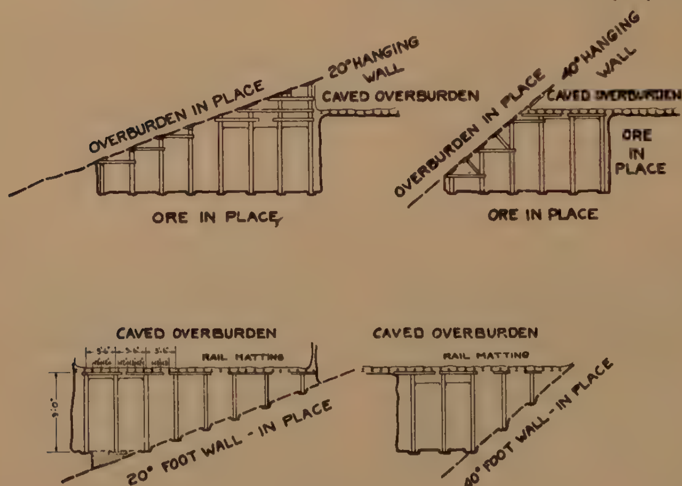

Fig 7

Fig 6 and 7—Top-slice mining details.

Large scale individual floor maps showing pilots, cross cuts, chute locations, ore contact with dip and strike and all assays are supplied to mine foremen. The position of the matting on the floor is also plainly marked.

matting rails laid in this way will cover 162 sq ft of floor area.

When "pulling" stopes to throw down the overburden the posts and headboards are pulled out one at a time, allowing the roof to cave. As the roof caves, the broken-

down overburden is sprinkled with water to ensure a consolidation of the caved material. If at first the roof does not cave readily, drill holes into the consolidated overburden are blasted around the edges to increase the area or span so that it will cave and fill the space completely.

After an area has been matted and caved, it is allowed to stand for two or three months to ensure consolidation before the next lower slice works underneath it. The time for consolidation will depend upon the character of the overburden, the amount of clay and fines present, and the amount of water sprinkled during the caving.

When the upper floor has been matted and caved, the pilot drives and crosscuts can be driven on the next lower floor. These galleries are parallel to the ones above but are offset about 10 ft between centers to permit raises with chutes and manways to lie between them, thus pilots on the odd-numbered floors are on one side of the raises, and on even-numbered floors are on the opposite side. The pilots and crosscuts of this lower floor are driven to the limits of the ore or to pillar lines, and consequently may extend beyond the matting of the first floor on the hanging-wall side, and into the footwall on the foot-wall side. The ore beyond the matting rails of the overlying floor is mined out first by small stopes driven at right angles to the pilots, using the same timber post and headboards as were recovered when pulling stopes on upper floor. When these margins are mined out, matted and caved, then the steel posts are used in mining underneath the mat.

When using steel supports, the standard stope unit is 10.5 ft wide measured along the pilot drive, and 16 ft long measured from the side of the pilot drive. A similar stope is cut out on the opposite side of the pilot drive. The pilot drive is 4 ft

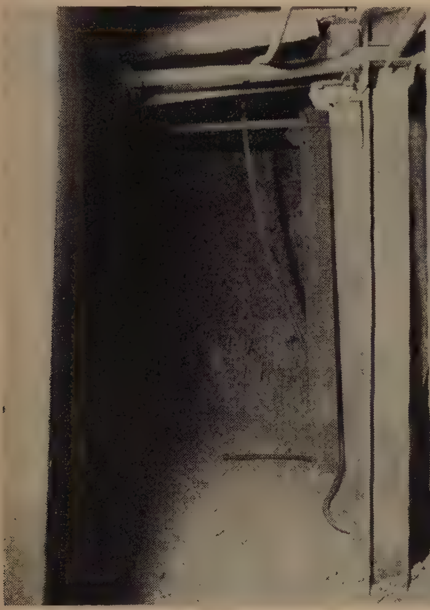

Fig 8

Fig 9

Fig 8—Steel posts and collar braces in position underground.
Fig 9—Steel posts, collar braces, and caps show against the fence line of stope.

wide between 2 stopes each 16 ft long. This gives 36-ft centers between pilot drives.

When starting a stope the roof of the pilot is broken down to expose the matting rails of the overlying floor for the width of the pilot, and a length of about 5 ft 3 in. along the pilot. In this space, two steel posts and one steel cap are set along each side of the pilot drive. These are connected up with collar braces and blocked securely and squarely in line with the pilot drive. The posts are set in a 4-in. hole in the floor. Each matting rail exposed in the roof is securely wedged up from the cap by soft wood wedges.

The standard steel post (Fig 4, 6 and 7) is made from a 5 by 6 in. beam weighing 25 lb per ft. The length is 8 ft 3 in. with a 10° angle cut on the top and a square cut on the bottom. The top is reinforced by riveting two pieces of angle iron to the web forming a smooth beveled top. Four angles are riveted to form supports for the collar braces 9 in. below the top. On the beveled top of the post is placed a 10° hardwood wedge which forms a level support for the steel cap. In setting up the post, it is important that the inclined top slopes down in the direction opposite to that in which the post will be pulled out (Fig 8 and 9).

The caps are cut from long rails weighing from 90 to 110 lb per yard. The length of a cap is 4 ft 7 in. and it weighs 145 lb for the 90-lb rail and 160 lb for the 110-lb rail.

The collar braces are made from 2 or 3 in. pipes, generally secondhand, cut 36 in. long with square ends and no threads. Near each end, holes are drilled through the pipes, and ¾-in. pipes are welded into them, one point projecting about 1.5 in. These collar-brace pins, when in vertical position, fit loosely into holes in the angle supports on the post and hold the post in an upright position and spaced 3 ft 6 in. centers in four directions. The posts are blocked tight against the collar braces and the end post in each line is blocked securely against the ore or waste.

When the first set has been securely blocked, the roof of the pilot drive is cut out to expose the matting for an additional length of 5 ft 3 in. to make room for the second set which is made up of one post and one cap on each side of the pilot drive, connected with collar braces, lined up with the center line and securely blocked. These two sets, one double-post set and one single-post set, form the unit stope width of 10.5 ft. The double set is always placed against the cave line where the roof weight is greater and the single-post set stands between the double set and the ore in place. Throughout the pilot drives, the double and single sets always alternate.

Standard practice is to raise two additional sets in the pilot drive before commencing to stope in order to leave room to set up the post-pulling equipment.

Fig 4 shows plan and section along *A-B*, where four sets are set in the pilot *A-B*. The fence lines *a b* and *b c* were placed in the previous stope before caving.

The ore face *a d* is drilled and blasted using *a b* as a free face. The two-post set against the cave line *a b* is erected as soon as the broken ore is removed and then the one post set, with all the collar braces in place, the end posts blocked against the ground, and the matting rails securely wedged up from the caps.

The stope face advances until three lines of three posts each are set up outside of the pilot drive. The remaining ore up to line *b c* can probably be recovered without setting additional posts if the rail matting in the roof is in fair shape. When the ore is all cleaned out, the fence *d c* is placed against the ore face to protect it from the waste cave.

The fence consists of 4-in. wood posts 9 ft long set vertically about 4 ft apart. Slab wood is nailed on the posts in such a manner that when pulling a stope, the

near ends of the slab wood will be on the stope side of the post for easy recovery.

After the fence *d c* is built, the matting rails are placed on the floor in position as shown by dotted line in Fig 4, and covered with 6 in. of clay from the cave. The stope is now ready for pulling (Fig 10).

The pulling equipment consists of an Australian Monkey Winch fitted with 30 ft of ¾-in. steel cable, a snatch block pulley and several ¾-in. short chains. This winch consists of a steel drum, one flange of which is a gear with ratchet teeth on the rim. The drum is mounted in a yoke which can be anchored to a post or cross timber outside of the stope to be pulled. A 5-ft pump handle rotates the drum which winds up the cable with a powerful pull. Three men on the handle will generally pull out a post, but if the roof weight is too heavy to permit pulling, then the foot of the post is undermined with an air pick which loosens the head sufficiently to pull it with the winch (Fig 11).

When pulling a stope, the winch is set up at point *e* (Fig 4) in the pilot *A-B*. The snatch block is fastened with a chain to the foot of the post at *f*. Two wood posts 7 in. in diameter and 8.5 ft long are set up at points *g* and *h* with headboards to hold the matting rails temporarily while pulling and recovering the materials beyond. The three steel posts are pulled out, one at a time, and the caps, fence material, and matting, are recovered by a boat hook. The wood posts are then pulled out and reset in the next space between steel posts. The pulling continues until the pilot drive is reached, and the fence set up at *a d*. The sets in the pilot drive are left standing until the stope on the opposite side is finished, as is shown by the dotted lines. Then all the sets are pulled out and a fence is built across the pilot drive at *d*.

While pulling a stope, the recovered material is stored in the adjacent crosscut, if available. If, however, no crosscuts are nearby, a space *l* is cut out and standard post and cap with collar braces are set up and the space used as storage.

A large hole is drilled in the web near each end of all rail in matting and in caps to permit easier recovery with a boat hook.

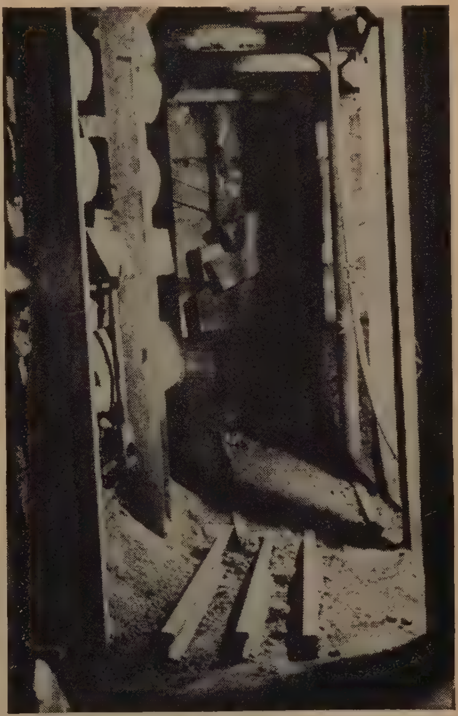

Fig 10—Steel rail floor "mat," posts and fence.

Plan and two sections of the ore body with a group of top-slice stopes from the third floor to the sixteenth floor in various stages of operation are shown in Fig 5. The chutes and manways are spaced 50 ft apart along the pilot drives. Haulage-level galleries are crosscuts on 50-ft centers with chutes and manways at 36-ft intervals along these galleries.

Ventilation foul-air-return galleries, located on the 5th, 13th, and 27th floors, are driven well in advance of the stoping operations, and placed so that they will serve as pilots when stoping approaches. The ventilation is then transferred to lower

floors. The ventilation raises are shown solid in the plan of Fig 5.

The haulage galleries are abandoned, stripped of tracks, timber, and other equipment, and are back filled to prevent crushing when the stopes approach within 2 or 3 floors. The ore is then passed through chutes to the next lower haulage gallery.

Experience to date indicates that the cave line should not be a straight line but a zig-zag line which averages about 50° diagonal with the long axis of the ore body. A spacing of 30 ft has been adopted between adjacent stopes on the same floor and a 60-ft minimum spacing between stopes on adjacent floors to ensure sufficient time for good consolidation of the caved overburden. The direction of the cave line will depend upon which side of the ore body is the softer or has softer walls. The softer or weaker ore should be mined first, leaving the harder or best standing ore to be mined last.

FIG 11—AUSTRALIAN MONKEY WINCH SET UP IN PILOT DRIVE WITH SNATCH BLOCK TO PULL POSTS FROM STOPE FACE NEAR BY.

The pilot drives are on the center lines of a series of long blocks 36 ft wide which are mined by the regular top-slice stopes from both sides of the pilot drives. The irregular margins on both sides of the ore body are mined separately and before mining the uniform center blocks. These marginal blocks are not all full height so that timber posts are used whenever there is not height for a steel post.

Fig 7 shows sections of hanging and

footwall marginal stopes. When working under the hanging wall, the stopes require additional matting rail but when working on the footwall, excess matting is recovered. If the dip of both walls were the same, then the additional matting required on the hanging wall would equal the excess matting recovered on the footwall stopes.

Referring to Fig 5, the matted area of any floor extends in a single layer from cave line to cave line in the direction of the long axis of the ore body, and from the hanging wall to the footwall across the ore body. There is no overlapping except temporarily when the matting rails are laid on the floor of a stope just before pulling the stope and before recovering the matting in the roof.

The matted area of any group of stopes on several floors is obtained on the maps by projecting the upper floor stope limits and its cave line, together with all the intermediate floors and their cave lines, to the elevation of the lowest floor. The cave line and limit of the ore body on the lower floor should also show on the map. By connection with a broken line, all the hanging-wall ends of the cave lines will give the hanging-wall side of the area and in a similar way, the footwall side of the area is shown. The upper and the lower floor ore limit are the two end lines of the matted area. This area in square feet divided by 162 gives the tons of matting rail in place, provided all the area has been matted. Any area not matted should be shown on the individual floor maps and deducted in the calculation. The reasons for not matting may be any of the following:

1. Proximity of the footwall, when the dip is flat (see Fig 7, footwall sec 20° dip). The ore is taken out to the footwall and the space filled with waste before setting up the steel posts.

2. Low-grade ore, or noncommercial ore, that will not be mined on the next lower floor unless the grade improves as shown by later development.

3. Suspending operations or delayed operations due to pillar lines limit, and other causes, to avoid tying up steel rail matting in idle parts of a mine.

Individual floor maps also show the position of stope faces at the beginning of each month and the area between these monthly stope positions is the area mined during the month. The division of monthly stoped area by the total matted area will give the number of floors mined out each month, and the position of the various working floors can be predicted for several years in advance. These figures are necessary in planning future work in preparing haulage and ventilation galleries, chutes, manways, and so on.

The average working cycle of two unit stopes on each side of the pilot as shown in Fig 4 is approximately as follows:

	Shifts	Men	Man Shifts
Raising pilot sets	2	3	6
Stoping right side	9	3	27
Pulling right side	2	3	6
Stoping left side	9	3	27
Pulling left side and pilot drive	3	3	9
	25		75

The tonnage in this double stope unit is 300 tons, calculated at 10.5 cu ft per ton and deducting the volume of the pilot which was previously driven.

While mining and timbering, 60 man shifts produced 300 tons, or at the rate of 5 tons per man shift. For the total cycle 75 man shifts produced 300 tons at the rate of 4 tons per man shift.

Cyprus costs and consumption of steel used in top slicing from 1935 to October 1939, were as given in Table 1.

Repairing Steel Posts

In the top-slice stopes, steel posts are occasionally bent by standing too long under a heavy roof or during the pulling.

These crooked posts are sent to surface for repairs. They are heated in an oil

furnace 10 ft long, pulled out to a cast-iron bench and straightened by two men with 16-lb. hammers striking on a flat tool

TABLE 1—*Costs and Consumption of Steel Used in Top Slicing*
Dry Tons Mined—1,796,136

Rail Matting	Long Tons	Value	£—$4.50
Total store issues	2,612.9	£21,036.65	$94,700.00
In place underground Oct. 1939.........	1,295.7	10,395.97	46,700.00
Lost...........	1,326.2	£10,640.68	48,000.00
Depreciation of steel in place, est. 50 pct....			23,350.00
			$71,350.00

$\frac{1326.2 \times 2240}{1,796,136}$ = 1.65 lb steel lost per ton mined.

$\frac{7,135,000 \text{ cents}}{1,796,136 \text{ tons}}$ = 3.98 cents per ton mined.

Rail Caps			
Store issues.....	387.5 tons	£3,065.26	$13,800.00
In place underground Oct. 1939.........	253.4	2,004.48	9,100.00
Lost...........	134.1	£1,060.78	$ 4,700.00
No depreciation			

$\frac{470,000 \text{ cents}}{1,796,136 \text{ tons}}$ = 0.262 cents per ton mined

Steel Posts (units)			
Store issues.....	2,041 posts	£5,047.54	$22,470.00
Usable underground Oct. 1939.........	1,756	4,342.74	19,300.00
Lost...........	285	£ 704.80	3,170.00
Depreciation of posts in use 50 pct........			9,650.00
			$12,820.00

$\frac{1,282,000 \text{ cents}}{1,796,136 \text{ tons}}$ = 0.715 cents per ton mined.

Summary of Steel Costs:

	Cents per Ton Mined
Rail matting..................	3.98
Rail caps......................	0.262
Steel posts....................	0.715
Total......................	4.957

held and guided by the blacksmith. By sliding or turning the hot post into various positions and directing the hammer blows, the post is straightened and allowed to cool slowly.

Each time a post is heated and straightened, it is given a chisel mark near the top. There are several posts in the mine which have nine chisel marks and are still in service.

The heating, contact with pyrites dust, and humid atmosphere underground, all cause considerable rusting and scaling of the steel which gradually reduces its weight with a consequent loss of cross-sectional area which eventually will cause failure. In the four years use of steel, we estimate a depreciation of 50 pct on the steel remaining in use.

We have noticed that, on the foot of some posts, web and flanges are worn very thin by corrosion of the ore and these have a tendency to bend over or curl up. This suggests that on orders for future posts, web and flanges should be thicker and the foot of the post should be reinforced by angles similar to those used on the top of the post.

Records of the amount of timber used in top-slice operations have not been sufficiently segregated from total timber costs to enable us to state average figures. The costs per ton mined exceeded the average costs for steel but the use of timber was gradually being reduced in prewar stoping operations and more of the timber was being recovered in unbroken condition, fit for use again. The principal use of timber in stopes was for "fencing" to separate the waste overburden from the ore. This timber was repeatedly recovered and used again for similar service. Another use of timber is for blocking end posts and fence lines as ore is extracted. When "pulling" stope posts there may be runs of the caved overburden that will necessitate erection of breast boards to protect the steel, but these runs seldom occur if the caved overburden has been properly sprinkled while caving.

Pilot drives and crosscuts require timber supports when the ore is not self-

supporting. When pilots and crosscuts are enlarged for steel supports as stopes approach, the timbers are taken out and replaced by steel after enlarging the pilot drives by shooting down the back of the pilot drive for a length equal to the width of the next stope to be mined. There is not much breakage of timber used in pilot drives and the timber usually can be used several times before it becomes necessary to discard it.

Cut-and-fill Stoping

In the upper, irregular, and comparatively narrow parts of the Mavrovouni ore body, and in much of the Skouriotissa ore body, stoping was done by cut-and-fill methods. Two variations of the methods were employed: (1) horizontal slice cut-and-fill, and (2) vertical slice cut-and-fill. In both types it is advisable to start the initial cuts not more than 40 ft below the top of the ore body to be stoped, in order to minimize cracking, crushing, and movement of the ore, which causes oxidation, heating, and sometimes fires.

In most of the Cyprus cut-and-fill stoping operations the waste for filling is obtained from raises driven into the roof or hanging wall. At the top of the raises the waste is undercut by drilling and blasting in order to cause caving of the waste.

Pilot drives are driven one floor above tramming level and usually at right angles. These connect with the waste raises and are connected with tramming levels at two or more places in order to give ventilation through the pilot drives and stopes. Connection to tramming levels include ore chutes for removal of ore. Stope units are cut 8 ft high, 10 ft wide, and about 46 ft long, on each side of pilot drive, making a stope block 100 ft wide. Each stope unit is then filled and back packed tightly with waste brought through the pilot drives and laid against the lagging along the side of the stope. When filling is completed new stope units are started alongside the completed ones and at same elevation. When the entire stope block has been mined out for a height of 8 ft the chutes and manways are extended upward, and the lower half of pilot drive is filled, leaving the upper half open to maintain ventilation and a small passageway for men. The roof of the pilot drive is then raised 4 ft by mining the ore. When this operation is completed the new floor of the pilot drive is raised 4 ft by filling, and roof is raised 4 ft by mining. This creates the pilot drive for the second horizontal slice, immediately above the first slice. This cycle is repeated until the entire block is mined out to the roof or hanging wall of the ore.

Vertical slice cut-and-fill stoping is essentially the same as the operation in the pilot drives of the horizontal slice cut-and-fill, omitting the unit stopes alongside the pilot drives. The ore in the roof of the initial cut of the first vertical slice is attached to and supported by the ore on each side. Ore in the roof of second and subsequent slices is attached to ore on one side and partially supported by the packed filling of the previous slice on the other side. Each cut is 8 to 10 ft high and as wide as the ore will safely stand. The roof is partially supported by posts set with small end down on footboards on the filling. When the floor of the cut has been filled for half its height and roof mined out for an equal height, the posts which are then half buried in waste, are pulled out and reset on new floor (filling) to support the new roof.

When one stope block closely approaches an adjoining stope block the remaining pillar of ore will have support only from waste filling on each side. This is insufficient for safe mining by vertical slice cut-and-fill. Such a pillar is usually extracted by top slicing. The stope opening in this case may be filled by pulling (throw-

ing down) the overlying waste, or by back-packing on top of timber-floor lagging set on top of ore pillar.

Waste may be moved from waste raises to point of filling by slushers, cars, or wheelbarrows.

Most of the chutes and manways (also airways) are built of precast concrete blocks, and carried upward in the filling alongside stopes so that they can be used again when making the subsequent adjoining stope.

References

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2. B. F. Tillson: Mine Plant. XXXV–10. (1935) Rocky Mountain Fund Series, AIME.
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Cyprus Mines Copper Again

By J. L. Bruce,* Member AIME

(Los Angeles Meeting, October 1947)

After six years of war-enforced idleness, Cyprus copper mines are operating again. This relatively long shutdown seems infinitesimal when compared with something like seventeen hundred years of inactivity that followed the important operations of the ancient Romans, and continued until the twentieth century. The historical records of the ancient operators are scarce but full of interest to those who realize that Cyprus copper production was important more than two thousand years ago. This paper, however, will give only a brief review of the ancient history and the evidence of ancient mining which have been quite fully covered in other publications.[1-6]

The purpose of this paper is to give a general sketch of the Cyprus Mines Corporation's enterprise in the Island of Cyprus, with special emphasis on those conditions and practices which are unusual. The picture is not intended to be complete, and many interesting details will be omitted. Some of these may be described in subsequent papers covering departmental practices.

The principal operations of Cyprus Mines Corp. are in Cyprus, where it has leasehold concessions from the Cyprus Government covering about 50 square miles. It holds additional areas under "Prospecting Permits." The consideration for the concessions is a royalty on production, coupled with minimum annual rentals.

Cyprus is a British Colony in the Eastern end of the Mediterranean with an area of about 3000 square miles and a population of about 400,000, of which 65 pct may be classed as Greeks on the basis of language, religious training, and traditions. About 25 pct are Moslems, principally of Turkish extraction. Prevailing languages are Greek and Turkish. A large proportion of the supervisory and administrative staff and practically all of the skilled and unskilled craftsmen and laborers are drawn from the native population. The "overseas" staff constitutes not more than 2 pct of the employees. The majority of these are from the United Kingdom, the British Dominions and Colonies. About one-third are from the United States of America.

The topography of the Island is rugged, with mountains and foothills covering more than two-thirds of the area. Between the Kyrenea range rising to 3000-ft elevation, along the north coast, and the Troodos range (maximum elevation 6400 ft) occupying the southwest half of the Island, lies a cultivated rolling plain, the "Mesaoria," dry in summer and watered only by the winter rains and by scanty irrigation from small mountain streams and wells in the spring and summer. Climate and natural vegetation are characteristic of semi-arid conditions. The "urban" population, about one-fourth of the total, lives in the five port towns, Famagusta, Larnaca, Limassol, Paphos and Kyrenia, and in the capitol Nicosia, near the center of the Island. The suburban population lives almost entirely in villages or small towns, with very few people residing in isolated country homes.

Manuscript received at the office of the Institute Oct. 24, 1947. Issued as TP 2459 in Mining Technology, September 1948.

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1 References are at the end of the paper.

The higher elevations of the Troodos range are dotted with villages and communities which serve as comfortable summer resorts for those who wish to escape the heat of the plains.

FIG 1—CRIBWORK OF ANCIENT ROMAN MINE SHAFT ENCOUNTERED IN UPPER LEVEL OF MAVROVOUNI MINE.

ANCIENT HISTORY

Cyprus was perhaps the earliest of the important producers of copper smelted from areas that contained the metal in combination with sulphur. It may have been among the earliest of those which recovered copper from ores containing other than metallic copper. Historical records of ancient copper production are surprisingly meager. It is known that the English word "copper," the Latin word "cuprum," and the Greek word "kupros" are derived from the name of the Island, which is more ancient than that of the metal. Cyprus copper metal undoubtedly was delivered to Egyptian kings before the Island was conquered by Thotmes III about 1500 B.C. Homer dressed Agamemnon in Cyprian armour, and Alexander the Great had a Cyprian sword—both probably of hardened Cyprus copper or bronze. The poet Stisichore (640 to 550 B.C.) mentions the Telchines, who are also referred to in the writings of Strabo, Diodorus of Sicily, and Nicolas of Damascus. Diodorus classifies them as metallurgists of Cretan origin who settled in Cyprus before going to Rhodes. Strabo says that they were the first who worked copper and iron. We do not know what date to ascribe to the Telchines. They were perhaps merely mythological characters, or they may have been living practical metallurgists.

In Cyprus there are many and extensive evidences of ancient operations in the form of antique surface and underground excavations and dumps, timbering, mining tools, and smelter slag piles. (Fig 1.) Pre-Roman slag piles, most of which are credited to the Phoenicians with some uncertainty, are characterized by high-iron and low-manganese content and somewhat higher copper content and lighter brown color with more weathered appearance, as compared with the "Roman" slags which usually are more dense and glassy and of distinctly darker color. This is largely the result of remarkably high percentages of included manganese that obviously must have been transported from outside the copper mining area for use as a smelter flux.

The Roman occupation of Cyprus in 58 B.C. followed 14 centuries of control by Egyptians, Phoenicians, Assyrians, Persians, and the Ptolemies in succession. Roman production was probably most active between 58 B.C. and A.D. 200, and production must have greatly exceeded that of all predecessors. The most detailed account of any of the ancient mining activities was discovered within the past 20 years by Dr. Joseph Walsh while translating the writings of Galen,

"the father of experimental physiology," and physician to Emperor Marcus Aurelius. In A.D. 166 Galen visited the mines for the purpose of collecting medicinal salts that were produced there. He described his observations in detail, and we may conclude that his description referred to the mine now known as the Skouriotissa. It seems probable that mining gradually declined and finally became inactive within a few generations after his visit. We find no historical reference to active mining for approximately 1700 years after Galen's visit.

FIG 2—RUINS OF BELLA PAISE ABBEY ON NORTH SLOPE OF KYRENIA MOUNTAIN RANGE NEAR KYRENIA.

MODERN HISTORY

From the Roman period until 1878 when Cyprus passed from control of the Ottoman Empire to that of Great Britain, interest in metal mining appears to have been entirely dead. Casual prospecting commenced in 1882 and substantial explorations were under way at Lymni at various times from 1886 to 1914, when the war caused suspension of activities which had yielded no marketable products except a modest quantity of "cement copper" precipitated from the mine waters and from leaching of ore stacked in dumps.

At the Skouriotissa mine shallow prospecting work had been done occasionally prior to 1913, but prospecting permits had all lapsed before February 1914 when Mr. Charles G. Gunther obtained a special prospecting permit. Mr. Gunther, supported financially by Seeley W. Mudd and Philip Wiseman, had set out in 1912 to investigate the sites of ancient mines in Near East countries. Eventually this led him to Cyprus where he was most impressed by the showings at Foucassa Hill, the site of Skouriotissa mine. Shallow manual exploration adits and shafts convinced him that it would be necessary to reach greater depths to find the ore. Churn-drilling was started in March 1914, and continued until November when the drillers returned to the United States on account of the war after drilling 20 holes with total footage of nearly 7000 ft. Commercially valuable ore was not encountered until the 9th hole. Drilling was resumed after the war in October 1919

During the war years, limited underground developments were continued. On November 20, 1915, the "Skouriotissa" mine lease was granted to Gunther, Mudd and Wiseman, who organized Cyprus Mines Corp. in 1916.

After churn-drilling was resumed in 1919 much time was consumed in establishing satisfactory market outlets for the

pyrites and in development of the mine, establishment of transport facilities, and procurement of equipment and housing accommodations for employees.

The Mavrovouni mine ore body was discovered by churn-drilling but not until after eight holes had been drilled. Continuation of the program until more than 35,000 ft had been drilled in about 50 holes clearly indicated the existence of several millions of tons of good cupreous pyrites ore of a character more amenable to concentration than that of the Skouriotissa mine.

Cyprus Mines Corp. has been producing and shipping cupreous pyrites products from Cyprus since 1922 with the exception of the period from June 1940 to May 1946, during which practically complete discontinuance of production resulted from shortages of supplies or shipping facilities caused by the war. Early shipments consisting of suitably crushed and sized crude ore about 2.25 pct Cu, 48.0 S, were made from the Skouriotissa mine ore body, the ore from which was not economically amenable to concentration on account of the high water-soluble content. Total shipments from Skouriotissa mine ore body have exceeded 2,700,000 long tons.

The Mavrovouni mine ore body is large and not greatly different analytically from the Skouriotissa ore body except that copper content averages about 4.25 pct instead of 2.25 pct. The average sulphur content is about 48.0 pct. The copper content of this ore is undesirably high for acid-works roasters, although a limited quantity would have been acceptable to those acid-works purchasers of cupreous pyrites "fines" who could mix it with pyrites containing much less copper, provided there was a satisfactory outlet for the cupreous cinders (calcine) resulting from the roasters. Realization of these restrictive limitations and of the fact that the Skouriotissa mine might be able to supply most, if not all, of the cupreous pyrites market requirements, caused Cyprus Mines Corp. to decide to install a concentrator for the purpose of separating as much as practical of the copper minerals from the iron pyrites of the Mavrovouni mine ore body. It was expected that some of the pyrites in the concentrator tailings could be marketed currently. The balance of the tailings was to be stored for disposition as a pyrites product at some indefinite future date. The first unit of the concentrator was put into operation in 1934. This had a capacity of about 25 long tons per hour. Additional units were added until a capacity of 85 long tons per hour was reached (in 1938). Total ore mined at Mavrovouni mine has been in excess of 2,500,000 long tons.

Of secondary importance is the production of "cement copper" from copper-bearing solutions derived from leaching operations.

During the depression of 1931 and 1932 when the market for cupreous pyrites was reduced to less than 100,000 tons per year, Cyprus Mines Corp. decided to provide work for its unemployed wage earners by using them on exploration and development of previously discovered occurrences of gold-silver ores of uncertain economic value. Sufficient ore was developed to justify shipments of some of the best crude ore, and eventually after much metallurgical research on this very unique and refractory ore, a cyanide plant was constructed in 1932. Total production up to cessation of operations in 1940 was about 81,000 oz Au and 480,000 oz Ag from about 160,000 tons of ore produced from the originally discovered ore body and from many small mines widely separated and scattered over the cupreous pyrites mineralized zones.

Gold-ore discovery and production from surface down to sulphide zone led to the discovery of the North Mathiata pyrites ore body. In similar manner a large cupreous pyrites deposit at Aplike was discovered. All other gold mines marked the outcrops of low-grade, disseminated cupreous pyrites or very small deposits of massive pyrite.

Geology and Mineralogy

Politically and sociologically Cyprus has had its "ups and downs." Few countries have been more subject to the tides of racial migration. At the borderline between East and West, between Egypt and Asia, and between Christian and Moslem, it has usually been the victim of more powerful influences. Its geological ups and downs have successively connected and disconnected it from the Asiatic mainland near Alexandretta. This is reflected in the fossil deposits which contain the bones of pigmy elephant and hippopotamus, isolated antecedents of their larger descendants on the mainland. The reservoir of underground fresh water on which Cyprus Mines Corp. is dependent for the large quantities required in concentrating is the result of subsidence of gravel-filled river beds and detrital fans to points below sea level. In these are collected waters of the rainy season and runoff from the mountains.

No attempt will be made to give more than a brief sketch of the general geology of the Island of Cyprus or the general areas adjoining the principal ore bodies (Fig 3). Sedimentary formations cover about 80 pct of the Island, igneous rocks the remainder.

Fig 3—General geological map of Cyprus.

The oldest sedimentary rocks form the ridge of the Kyrenia Mountains which lie within three or four miles of that part of the north coast which extends eastward from Morphou Bay. These are generally considered to be Cretaceous. This range of mountains contains only small bodies of igneous rocks and no noteworthy indication of ore mineralization. Surface exposures of the igneous rocks of the Island reach the coast line only along a five mile stretch between Khrysokhou Bay and Morphou Bay (Fig 4). From this their axis extends southeasterly towards Cape Kiti, 8 miles south of Larnaca, as a strip of igneous rocks about 17 miles wide, almost surrounded by sedimentary rocks which usually lie over and dip away from the igneous rocks. Almost all, if not all, of the pyritic and the gold-silver ore bodies (which have all been derived from massive or dis-

seminated cupreous pyrite deposits) are confined to the relatively narrow rim, seldom more than 3 or 4 miles wide, of mountain mass of diabase and serpentine. This rim surrounds the diabasic mountain mass with the exception of a strip about

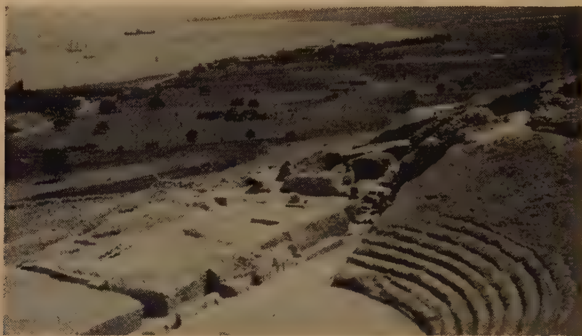

FIG 4—LOOKING EAST: MORPHU BAY, KARAVOSTASSI VILLAGE AND CYPRUS MINES CORP. JETTY FROM RUINS OF ANCIENT GREEK THEATRE EXCAVATED BY SWEDISH ARCHAEOLOGICAL EXPEDITION.

5

6

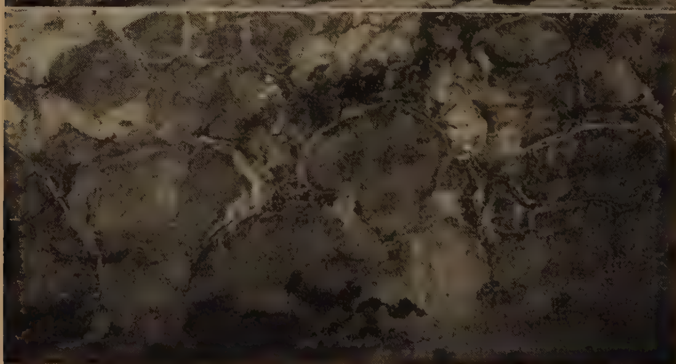

FIG 5—ERODED PILLOW LAVAS ON SURFACE. HILL ABOUT 30 FT HIGH.
FIG 6—UNDERGROUND EXPOSURE OF PILLOW LAVAS IN FACE 7 FT HIGH BY 12 FT LONG.

extrusive spilitic or andesitic lava rocks, largely pillow lavas (Fig 5 and 6), which lie as a rim on or against the principal 8 miles long on the northwest coast of the Island between Krysokhou Bay and Morphou Bay (from which it has appar-

ently been removed by erosion, exposing the underlying diabase rocks) and another strip about 45 miles long on the south side of the mountain mass.

Relatively small areas of lavas are exposed near Trimiklini and Perapedhi midway along this south side, but it is reasonable to assume that more lava occurs underlying the Miocene and older sedimentaries on all sides of the diabase. It has been fairly well established that most of the lavas were formed under the surface of the sea, then elevated and eroded and again depressed below sea level before the overlying Oligocene and Miocene sedimentaries were deposited. It is probable that these were elevated and eroded and again depressed before the Pleistocene and Pliocene sedimentaries were deposited.

Geological studies have not yet demonstrated conclusively that the extrusives are younger than the latest of the diabase. Cullis and Edge[7] state: "We are inclined to think of the diabase as a good deal older than the other two igneous types. It may be early Tertiary, or even pre-Tertiary in age and is the source of the igneous material in the Kythrean sandstones." They also state their belief that: "The lavas welled up through fissures in the diabase or other formations in the course of the movement of depression during which the Idalian (Miocene) chalky marls were deposited. Their age is definitely Miocene. Pillow lavas of this type, associated with deep water marine sediments, have been shown in other parts of the world to be characteristic of such sinking movements of the crust." Cullis and Edge tentatively inferred that the serpentines were of later date than the lavas.

The serpentine rocks contain no noteworthy occurrences of base metals or pyrite. Occurrences in the diabase are infrequent and considered to be too small to be of commercial value. These perhaps lie in channels which may have fed ore deposits in overlying extrusives now removed by erosion.

The number of occurrences of massive or disseminated cupreous pyrites and of gold ores derived from cupreous pyrites, and their scattered distribution throughout the extrusive lavas, and their confinement to these, indicate the probability that this is the only favorable host rock. The reason for this is not yet clear. The absence of important mineralization in the diabase, and of recognizable mineralization fissures in the overlying sedimentaries is especially noteworthy.

The frequency of occurrence of dark brown umber (exported in the form of terra umbra) in the marls at their contact with volcanic rocks, and its occurrence at the base of the Miocene capping of the Skouriotissa ore body raises the question whether the umber occurrences were genetically related to the pyrite mineralizing solutions and geographically related to the mineralizing channels. Most terra umbra contains considerable manganese oxide in addition to iron oxides. At many of the occurrences there is no evidence of pyrite mineralization and the formation of umber is believed by some geologists to have been earlier than the deposition of ore minerals and perhaps cogenetic with the deposition of the marls on the volcanics. The marls and umber may both be later than the pyrite. The umber at Skouriotissa extends much beyond the margins of the pyritic ore body. It contains several percentages of manganese in most places, and practically no copper, gold, or silver. It is not easily distinguishable from the limonitic bands of oxidized pyrites that overlie the pyrite ore body in many places. The limonite also is practically free from copper, gold, and silver but low in manganese. It is not plentiful, as the prevailing oxidation product of the pyrite at Skouriotissa mine is a noncupreous, yellow, basic hydrous iron sulphate—principally raimondite or jarosite or a mixture of these. These will be

called raimondite for convenience and following established nomenclature.

The raimondite constitutes the overburden, or capping and in places the host rock for the unique "devil's mud" gold-silver ores. Presumably these were formed by meteoric waters by leaching of the cupreous pyrite with acid ferric sulphate yielding iron sulphate minerals, raimondite, jarosite, fibroferrite, romerite, copiapite, and others, with precipitation and concentration of the gold-silver at the points where the ferric solutions have been reduced to ferrous. This appears to be a conclusive illustration of the occurrence of "secondary" gold. The copper which must have been carried by these solutions has not been found with the iron sulphate minerals, the gold-silver ore, the underlying highly leached volcanics, or the underlying unaltered andesite. Its almost complete disappearance has not been accounted for satisfactorily.

All commercially valuable gold-silver ores found in Cyprus have been geologically derived from cupreous pyrites mineralization in the andesitic lavas. Values are found from the surface to the sulphide zone over pyrite deposits of massive, stockwork, and disseminated types. Ratio of silver to gold is usually 8 or 10 to 1. Very little visible gold is disclosed by panning but gravity concentrates contain relatively high gold and silver associated with dark heavy minerals which we have not identified. These may be iron oxides and in some cases oxide-coated pyrites. Much of the ore occurs within a few feet above the sulphide zone in the characteristic devil's mud.

Some of the values lie in the weathered and disintegrated, iron-stained, and sometimes silicified cap rock of the outcrop. This probably includes the residues of devil's mud from which the customary water-soluble contents have been removed by leaching action.

Cyprus Mines Corporation's gold-silver ores have come from 4 or 5 sizeable deposits and 15 or more small occurrences. All ores were hauled to the cyanide plant at Xero for treatment. Mining at North Mathiati disclosed an underlying pyrite body with fair sulphur, but low copper content. At Apliki the gold mining disclosed a fairly large body of cupreous pyrites of good grade and favorably located. At South Mathiati only disseminated cupreous pyrites was found. At other places no important cupreous pyrites was found. Most of the gold-silver ores, with the exception of those at Skouriotissa, were mined principally by open-cut methods. Depths exceeding 100 ft below surface were exceptional.

There is no evidence that the ancient miners recognized the existence of gold and silver in the outcrops or in the devil's mud. Their failure to do so probably resulted from the fact that so little of the gold can be detected by panning.

The pyrite ore bodies include massive bodies, stockworks, and disseminated deposits of iron pyrite in occurrences of irregular size, shape, and direction with varying amounts of copper, and small amounts of zinc, gold and silver. Arsenic, antimony, lead, nickel, cobalt, and selenium are well below commercially low limits. Other metallic impurities occur only in traces. The only important primary copper mineral is chalcopyrite. In many places alteration has progressed to considerable depths, however, and the secondary copper minerals, bornite, covellite and chalcocite are abundant. In the uppermost hanging-wall portions of some ore bodies the secondary minerals may account for as much as 90 pct of the copper, in the lowest footwall portions as little as 10 pct.

At the Skouriotissa mine almost all of the ore body is above the natural water table. It is intensely sulphated and impregnated with sulphates of copper, iron, and zinc, including chalcanthite, brochantite, melanterite, and various ferrous and ferric sulphates such as romerite, coquimbite, fibroferrite, copiapite, raimon-

dite, jarosite, and others. In places there are occurrences of limonite and specimens of alunogen and a leathery form of asbestos.

At Mavrovouni mine much of the chalcocite is the extremely fine powdered variety known as "sooty" chalcocite. Covellite and bornite very frequently occur as very thin films coating chalcopyrite and pyrite. In freshly broken unexposed ore there has been no sulphating, but sulphating to brochantite occurs rapidly on exposure to air. These conditions make it impossible to separate normally clean copper minerals from iron pyrite low in copper content by flotation, even with fine grinding of flotation heads or middlings.

Zinc occurs as sphalerite and as goslarite. Copper carbonates, silicates and oxides are seldom found even in the form of specimens. Limonite frequently occurs at the hanging wall of the ore bodies. This occasionally carries sheets and stringers of metallic copper. Gangue minerals of the massive pyrites ore bodies include small amounts of silica and alumina, usually the leached residue of the lavas. There are occasional occurrences of clay, gypsum and asbestos.

At some of the gold-silver mines and at the smaller pyrites occurrences mineralizing channels are recognizable as fissures which in many places contain little or no vein quartz or vein gangue minerals. These fissures show hydrothermal alteration and alteration by meteoric waters in varying degrees. The outcrops consist of lava rock, iron stained, and in some cases slightly silicified along the fissures or shear zone and for some distance away from the recognizable shearing. The silicification results partly from chalcedonic deposits. The iron staining results from oxidation of disseminated or massive pyrite in situ, and also from the oxidation of migratory iron sulphate solutions.

Exploration

Over large areas adjoining the large pyrites ore bodies and in some areas where pyrite ore bodies are not known, details of structural control are masked by extensive hydrothermal alteration of the poorly consolidated lava rocks. In pyritized areas the meteoric waters bearing iron sulphates and sulphuric acid in many places have caused much alteration. These conditions make it difficult to work out features of structural control and general patterns have not been well established. Fissures and shear zones do not persist for long distances. They may result from anticlinal or synclinal folding. While evidences of ancient workings were plentiful, there was little or nothing to indicate whether ore bodies had been exhausted. All old shafts and tunnels had been closed by caving and erosion and old dumps and subsidence areas had been so thoroughly eroded and disintegrated that they blended with the landscape except for the widely distributed appearance of oxidation and iron staining. There were very few indications of copper.

An example of the extent of obliteration by erosion is furnished at the eastern end of Skouriotissa mine, where Cyprus Mines Corp. underground development work encountered an ancient horizontal gallery about 6 ft wide by 6 ft high in country rocks at considerable depths below surface. Survey showed that this gallery continued eastward for more than 600 ft to where it was caved at a point near surface of the steep hillside. Examination of the surface indicated no sign of any dump or opening. Undoubtedly the gallery was driven from surface as an adit but the large dump had been completely washed away and the portal of adit had caved and had been filled by debris from the hillside.

Exploration methods included geological examinations, subsurface work by cuts, tunnels and shafts, churn drilling, underground exploration with auger drills in the soft hydrothermally altered lava country rock in Mavrovouni mine, geophysical surveys by resistivity and induced potential methods by Lundberg and Sunberg, self-

potential surveys by Broughton Edge and our own staff engineers, reconnaissance examinations by aeroplane, and pursuit of gold-silver values and raimondite occurrences found in stained outcrops. Recent work includes surveys made with the Heiland gravimeter and with the magnetometer. Anomalies are investigated by drilling or by shafts and underground galleries. Close investigation of some of the most promising outcrops demonstrated that they were artificial. An interesting example of this at Mavrovouni mine was a very puzzling iron gossan "outcrop" on the ridge of a hill which appeared to consist principally of Pleistocene clay. Trenching in this large and prominent outcrop disclosed interesting gold-silver values. This led to the sinking of a shaft which passed through the outcrop at relatively shallow depth into clay. Further study led to the conclusion that the outcrop which covered an area of something like an acre, was actually the thoroughly oxidized remains of a Roman or Phoenician stockpile of pyrite ore, or of low-grade material cobbed off and rejected before smelting. This was several hundred feet from smelter slag heaps.

In other places promising gossan outcrops on close inspection showed small charcoal enclosures demonstrating that they were actually weathered smelter slags. Some of the underground "horses" of waste rock in ore were discovered to be strongly reconsolidated waste filling as they contained small pieces of wood.

Fig 7—Looking east: Foucassa Hill in central background, with ancient Roman and Phoenician smelter slag piles showing as black hills at foot of Foucassa. Cyprus Mines Corp. offices, staffhouses, and clubhouse, between observer and slag piles. Skouriotissa mine underlies left flank of Foucassa Hill.

Churn drilling at Mavrovouni mine gave the erroneous impression that most of the ore body was of the disseminated type containing considerable silicious gangue rock. Most of the holes were cased down to the ore and it has been concluded that the fine marls and clays in the overburden must have been washed into the holes by water entering holes from outside the casings.

Skouriotissa Mine

This mine, which lies principally above ground-water line under prominently outstanding Foucassa Hill, was probably the largest of the ancient producers (Fig 7). Great piles of Phoenician and Roman smelter slags lie at the foot of the hill. Ancient openings to the mine had been obliterated by caving and erosion. Prospecting with shafts and tunnels failed to rediscover the large ore body, which was

subsequently found by churn drilling only after seven blank churn-drill holes had been drilled.

The ore body is flat-lying, lenticular, roughly elliptical in plan, about 2000 ft long and up to 600 ft wide, tapering from a few feet thickness at the edges to a maximum of about 150 ft near the center. The roof or hanging wall in many places is flat, nearly horizontal. In places it is much steeper. The floor or footwall slopes from all sides towards the center. There are no clearly recognizable "roots" or feed channels. A few fissures exist but these are unmineralized or only slightly mineralized. Three or four postmineral faults cause ore displacements ranging from about 25 ft to a maximum of a little over 100 ft. The westernmost 75 pct of the ore body is faulted downward about 100 ft with respect to the eastern portion.

The ore body is overlain entirely or almost entirely by well-drained and dried Miocene clays, marls, and limestones. The cover of overburden varies from about 50 or 75 ft to a maximum of about 200 ft. The width of the ore body in all places is much too great to permit any plan of mining which will avoid subsidence resulting from shrinkage of the filling.

The higher, well-drained and dried portions of the floor or footwall provide reasonably good foundations for timber or masonry mine supports. The lower portions consist of badly decomposed and disintegrated lava rock thoroughly saturated with extremely corrosive acid cupreous ferrous and ferric salts. Copious crystals of melanterite and other sulphates form on evaporation of the solutions and create very difficult problems in development, drainage, rock drilling, tramming, and other operations. Much of the ancient production came from simple adits, galleries and small open stopes, most of which had been filled by the ancients. Our earliest modern mining was confined principally to production from networks of simple galleries on many horizons not far apart vertically. This practice gave relatively cheap ore but it was carried to a point which caused fracturing of the ore blocks. This, in turn, resulted in

FIG 8—FAN STATION AT COLLAR OF NO. 7 INCLINE SHAFT.

oxidation and heating which progressed much too far before this mining method was supplanted by cut-and-fill stoping. This was applied first as "horizontal slice" cut-and-fill which subsequently was largely replaced by a specially developed "vertical slice" cut-and-fill system. In the latest operations both methods were being replaced in favorable places by the unique top slicing method developed at the Company's Mavrovouni mine and described in a separate paper.[5] Caving methods of mining and shrinkage stoping are precluded by the certainty that friction and oxidation would cause prohibitive heating.

The amount of heat generated by oxidation in the Skouriotissa mine is remarkable. All ventilation is by large electrically-driven exhaust fans (Fig 8). The exhaust fans discharge approximately 300,000 cfm at temperatures ranging from 106 to 110°F Ventilation is well regulated, however, and the temperatures in working places seldom exceed 85°F except in a few dead-end galleries which occasionally must be driven for ventilation connections. Most of the heat is picked up by the air in the return-air galleries which have been driven

in the unmined pillars and sections of the ore body to provide entirely independent passageways for the contaminated air from the stopes. The pyrites is hygroscopic, automatically absorbing moisture from the air and thus preserving a low humidity in the ventilating system. Almost invariable practice at both of the Company's pyrites mines is to carry both air-inlet and air-return galleries to all sections of the mine. Air-return passageways are seldom used for tramming or for any purpose other than ventilation.

At the Skouriotissa mine the filling for cut-and-fill stopes was drawn from raises driven upward into the Miocene overburden. These eventually developed into glory holes from the surface. It was seldom necessary to drill and blast in the glory holes as the overburden usually breaks into small pieces. Occasionally it was necessary to blast large pieces of limestone that caused choking of the raises.

Ore and waste are handled in 3½-ton (long tons) rocker dump cars hauled by 6-ton Westinghouse storage-battery locomotives. The high water-soluble content of the ore makes it quite impractical to use water drills. For protection against dust the men use masks with wet sponges while drilling.

Mavrovouni Mine

Evidences of ancient mining include large amounts of slag scattered over many acres. Most of this is of the Roman type. Some appears more likely to be of earlier age. It is not improbable that a part of the smelter ore was brought to this smelting site from other mines, which may have been a few miles away. Near the slag piles is an iron-stained outcropping extending over an area of four or five acres. There was little evidence of ancient shafts or tunnels, but the topography suggested eroded subsidence or open-pit workings much too small to account for all the slags. This outcrop in pillow lavas is separated from the nearby Miocene and Pleistocene sedimentaries by erosion detritus which perhaps covers part of the outcrop. The outcrop is unimpressive and had been pronounced unpromising for a large ore body by eminent geologists. Notwithstanding this a churn-drilling program was undertaken. This resulted in the discovery of the large ore bodies now being mined.

Mavrovouni mine ore body is a large irregularly inclined chimney which grows much smaller and lies flatly inclined to the north, as it rises from sea-level elevation. Contours and sections of the cupreous pyrites ore body and of the "fourth" grade relatively noncupreous ore body are shown in Fig 9 to 11.

This ore body is almost completely surrounded by the andesitic lava which has been hydrothermally altered to a considerable degree for hundreds of feet on all sides of the ore. The ore body contacted the sedimentaries (Miocene overlain by Pliocene and Pleistocene) only in one small area where a small chimney of ore rose from the main ore body and spread out as a narrow finger about 350 ft long and 50 ft wide under the sedimentary contact. The overburden of the ore body ranges from 400 ft to 750 ft in thickness and consists of brecciated lava overlain by Miocene and Pleistocene sediments.

At most of the contacts between the heavy sulphide ore (Fig 12) and the relatively unmineralized lava, much of the hydrothermally altered, considerably leached and disintegrated country rock is in a plastic or semiplastic condition. It is consequently difficult or impossible to determine whether obvious ground movement has been due to persistent faulting or to local ground adjustments caused by shrinkage during formation of the ore body, or during the progress of secondary enrichment. The western border of the ore body cuts off so sharply as to suggest that this may mark a major premineral or postmineral fault. Plenty of gouge exists at the

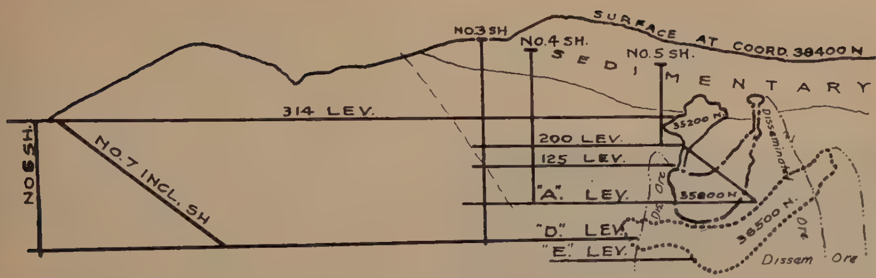

FIG 9

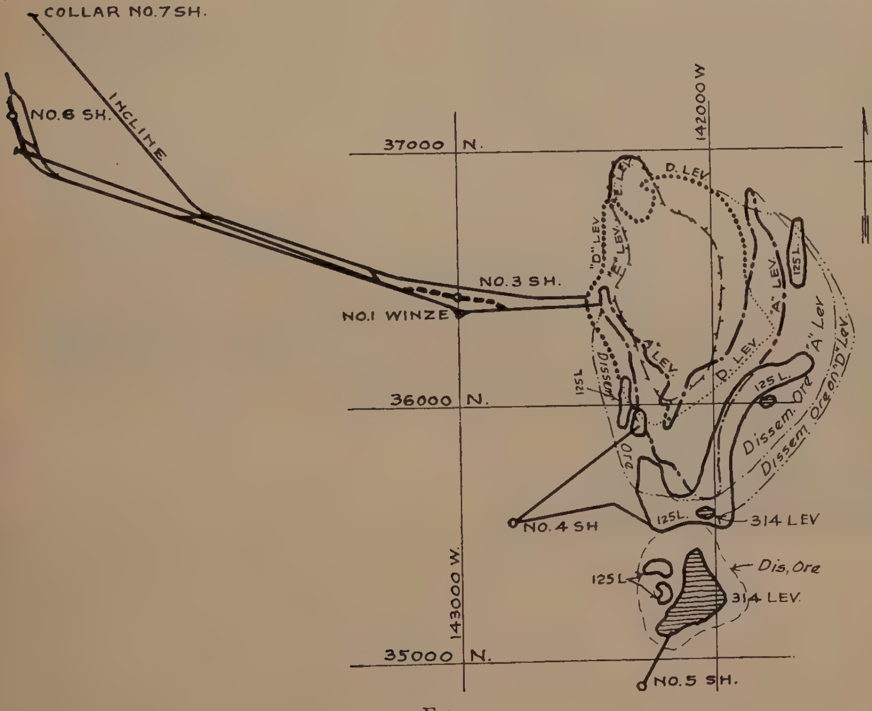

FIG 10

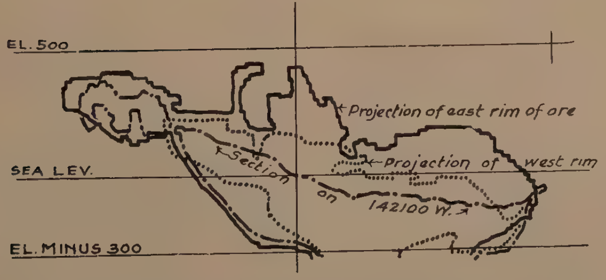

FIG 11

FIG 9–11—PLAN AND PROJECTIONS OF MAVROVOUNI MINE SHAFTS, WITH OUTLINES OF ORE BODIES SHOWN ON MAIN LEVELS AND ON SECTIONS.

boundary but no other persistent structural feature has been satisfactorily identified.

Only one major postmineral fault has been identified in a fairly convincing man-

FIG 12—LOOKING SOUTH: MODEL SHOWING HORIZONTAL OUTLINES OF MAVROVOUNI MINE HEAVY SULPHIDE ORE BODY ON SEVERAL LEVELS. CHURN-DRILL HOLE LOCATIONS ARE SHOWN BY THE VERTICAL RODS, TOPS OF WHICH ARE AT SURFACE ELEVATIONS. THESE RODS ARE USED TO SUPPORT THE HORIZONTAL SHEETS WHICH INDICATE OUTLINES OF ORE BODY.

ner. This may possibly be premineral. It strikes east and west and dips north with normal throw, separating the relatively small south ore body from the main mass. The displacement is about 75 ft. At contacts between heavy sulphides and lava it has most of the characteristics of a postmineral fault. Within the characteristically broken and brecciated mass of the ore body where ore lay on both sides of the plane of the fault (ore faulted against ore) there was no recognizable fault or fissure. This may have been obscured by those processes—secondary enrichment, mass shrinkage, crystal disintegration, and reconstruction, reconsolidation and recementation—which caused the characteristically brecciated appearance of most of the ore, especially that which has undergone secondary enrichment.

The immediate lava overburden of the ore body constituting the roof or hanging wall, generally speaking is notably different in appearance from the underlying floor or footwall. Both are lava, frequently vesicular. The overburden is usually well drained and dried and characteristically stained with tan, brown, and slightly reddish shades indicating the presence of ferric minerals. It usually breaks naturally into moderately small blocks and pieces which consolidate fairly well under pressure. When pulling the overburden in top-slice stoping this consolidation is assisted by sprinkling.

The footwall lava is characterized by its highly disintegrated, relatively moist, semiplastic condition and plentiful occurrence of gray and greenish-gray staining indicative of the presence of ferrous minerals. In places it also shows large and small contrasting blocks with ferric staining. When freshly developed, the ore body and the hanging-wall and footwall lavas contain few noteworthy oxidized or sulphated copper or iron mineral except the limonitic bands that frequently mark the boundary between overburden and heavy sulphide ore. In a few places the limonite contains sheets and crystal deposits of metallic copper.

The ore body exhibits extensive secondary alteration of the copper to chalcocite, covellite, and bornite. Much of the chalcocite occurs in extremely fine particles as sooty chalcocite ranging from a few microns downward in diameter. The covellite and bornite occur extensively as very thin coatings on pyrite and chalcopyrite. Chalcopy-

rite is the only primary copper mineral of importance. At the lowest level of the mine, 250 ft below sea level, 90 pct of the copper is in the form of chalcopyrite.

FIG 13—YARDS AT COLLAR OF NO. 6 SHAFT AND NEAR PORTAL OF 314 TUNNEL, MAVROVOUNI MINE. YARD HANDLING OF MINE CARS IS BY GRAVITY.

The degree of alteration of chalcopyrite to the other copper minerals is progressively greater in development galleries as they proceed from footwall to hanging wall, and as higher elevations are reached. Copper has not migrated far during alteration and there are no pronounced persistent zones of secondary enrichment or impoverishment. There are localized variations in copper content from about 1 to 10 or 12 pct or more, in bodies of sizeable tonnage, but somewhat similar variations in chalcopyrite content are recognizable and consequently it is not necessary to assume secondary migration as an explanation.

On exposure to the air the ore oxidizes by sulphating with unusual rapidity until protected by the sulphated coating which forms on the faces of the unbroken ore, or by exhaustion of the oxygen in the available air. In mining it is necessary to avoid pockets, such as dead-end raises in ore, in which the air will become impoverished of its oxygen making the area unsafe, or to bar off such sections against ingress of the workmen. At an early date it was recognized that oxidation progressed quickly in piles of broken ore until the oxygen was exhausted from the entrained air. This indicated the advisability of arranging the transportation of ore from stopes to concentrator so as to avoid long exposure to freshly entrained air (Fig 13). This was an important factor controlling the decision to eliminate skip pockets and mine-storage bins, which call for extra dumping and longer storage. Another principal reason for using cars instead of skips was to make it possible to more easily segregate several different types of ore.

Copper oxidizes first to brochantite soluble in acid but not in water. This changes to the water-soluble chalcanthite only after the ore reaches an advanced state of oxidation. The reason for this is not clear. Unfortunately it means that it is not prac-

tical to remove more than a small part of the sulphated copper by water washing before flotation.

Unusual features of Mavrovouni mining are described in some detail in a separate paper.[5] These include: (1) Topslicing with steel "matting," steel posts, caps, and so on. (2) Use of precast concrete blocks and poured concrete in galleries, shafts, and ore chutes of cylindrical shape. (3) Semi-automatic gravity handling of cars in hoisting and yard-transport systems. (4) Mine-ventilation system. (5) Fire refuge stations. (6) Incline exits with stairways.

Ore Transportation

Transportation of ores from Skouriotissa and Mavrovouni mines to the crushing plant and concentrator at Xero is by means of narrow gauge industrial railway operated by the Company. The railway line and rolling stock are somewhat unusual on account of the heavy loads—up to 35 tons per car—carried on 30-in gauge. Maximum weight of steam locomotives is 50 tons, maximum weight of rail is 90 lb per yard. The best installation, from the Mavrovouni mine to the crushing plant at Xero, is laid on steel sleepers. The most economical operation is with Plymouth diesel-engine locomotives (Caterpillar engines) purchased in recent years. These consume only one tenth of the amount of fuel oil (diesel oil) per ton of ore hauled that is required for the steam locomotives.

The selection of 30-in. gauge track and equipment was imposed on Cyprus Mines Corp. by Cyprus Government regulations instituted during earliest years of operation, in order to assure equipment and trackage which could be used in conjunction with the existing Government railway. Fortunately all loaded cars move on the down haul and only empty cars on the main line up-grade. Costs of transport have been surprisingly low under these circumstances. Haul from the Skouriotissa mine to Xero is about six miles, that from Mavrovouni about three miles. Prewar costs for the latter were less than three cents per long ton.

Gold ores from all the widely separated gold mines except Skouriotissa were hauled in trucks of less than 1½ tons rated load capacity, as Government paved roads and bridges were not constructed for heavy traffic. Much of the gold ore was hauled in excess of 45 miles to the cyanide plant located at Xero. Frequently the traffic of company motor vehicles exceeded 300,000 ton miles per month.

Research

The unusual character of the cupreous pyrites ores, and of the gold-silver ore, called for diligent and continuous research programs. First attempts were sometimes misleading, and it was not until several months after the first galleries were driven in the Mavrovouni mine ore body that it was realized that there was a very considerable oxidation of cut samples between the time of cutting and the time of analysis. The mine was then resampled at fresh faces and samples were put into vessels containing water at the point of sampling to prevent oxidation. By this method we were soon able to predict quite accurately that freshly broken ore—including a normal proportion of ore faces that had been exposed during mine development—would reach the concentrator sampling section with approximately 0.5 pct of the copper content in the form of chalcanthite, the water-soluble copper sulphate, while approximately 7.0 pct would be in the form of brochantite—a basic copper sulphate soluble in sulphuric acid but not in water.

Studies of the nature of the Mavrovouni mine ore reserves included a sampling and analytical program to determine the location and quantity of the primary copper mineral, chalcopyrite, as distinguished from the secondary copper minerals, bornite, covellite, chalcocite, and the sulphated minerals. The purpose of this was to deter-

mine what percentage of the copper could be extracted by acid ferric sulphate leaching or by heap leaching. Results were mapped and this has given us an unusually good picture of the results of secondary enrichment in this particular ore body.

Investigations included spectrographic analysis of ore and concentrates, microscopic examinations of polished sections of ore and concentrator products, and careful microscopic and analytical examinations of screen sizes. Fine screen sizes were carefully hand sorted with the aid of the microscope, and it was determined by this method that the cleanest pyrites that could be selected without any visible attached copper minerals still contained 0.20 pct copper and a considerable proportion of the normal gold and silver assays.

Before deciding to build a concentrator it was advisable to study possibilities of treating the ore by other methods including heap leaching, acid ferric sulphate leaching with electrolytic precipitation, semipyritic blast-furnace smelting with production of brimstone by Orkla process, and a number of variations or combinations of such processes. Consequently much of this research was conducted concurrently with development of best flotation practice.

Concentrator research commenced on the supposition indicated by churn drilling that most of the ore was disseminated with quartzy gangue, much of which we expected to reject in coarse size by gravity concentration on jigs and tables. This assumption was abandoned when mine development showed relatively small quantities of this class of ore. Fortunately this was disclosed before any mill was constructed. After laboratory flotation methods had been developed, a pilot-plant unit with capacity of 100 tons of ore per day was built. This operated for a few months while mine was under development. It enabled us to work out all essential elements of the flowsheet. During early stages of concentrator operations we built a complete flotation unit with grinding mill, classifier, and rougher and cleaner cells, with capacity to treat one ton of ore per day. This has been used frequently but intermittently during several years whenever it seems advisable to try out new reagents or new practices without disturbing current concentrator operations. It is also used whenever we wish to make tests on special types or grades of ore from the Mavrovouni mine, or from other ores which are not being delivered to the concentrator, or in making tests on Mavrovouni ore mixed with other such ores. Installation of a similar unit in any large mill treating complex or difficult ores is recommended. Flotation research included test-plant and concentrator flotation of deslimed ore and of the slimes separately, but this failed to show commercial advantages. Prolonged flotation of rougher tailings under varied conditions failed to give noteworthy recoveries of concentrates of sufficiently good grade to show additional profits. Additional finer grinding of concentrates, middlings, and tailings failed to give a profitable combination of recovery and grade of concentrates. These tests did indicate, however, that it would be profitable to improve the grade of concentrates and lower the assay of tailings by removing a middlings product assaying 3.0 to 5.0 pct Cu, for sale and shipment as cupreous pyrites, notwithstanding some sacrifice in the percentage recovery of copper into copper concentrates. A large proportion of the copper content of tailings was contained in the slimes under five microns diameter. A series of flotation tests on slimes separated from tailings failed to give substantial recoveries in commercial grade concentrates.

Various products of the concentrator were at times diverted to standard size Wilfley tables to determine possibility of beneficiation. It was concluded that any beneficial results could be obtained more cheaply by flotation or by other methods of gravity separation such as classification

by Dorr classifiers, including bowl type, or by hydroseparation.

Efforts to separate the zinc as a zinc concentrate were unsuccessful, as might be expected, since all zinc was activated by the relatively large amount of soluble copper sulphate

Research on ore included percolation leaching of mine-run and crushed ore and agitation leaching of finely ground ore, using water, and cold and hot acid ferric sulphate solutions. Tests were made under a great variety of conditions in the laboratory and in pilot plants on semicommercial scale. Heap leaching of large heaps on prepared floors was carried on for many months. Leaching by percolation in lead-lined tanks with hot acid ferric solution and agitation leaching in rotating cylinders furnished solutions for an electrolytic-copper pilot plant treating one ton of crude ore per day. This was first operated with graphite anodes in order to obtain high anode efficiency, but these anodes disintegrated rapidly, indicating heavy expense for anodes and further testing was done with lead and silver-lead anodes. Copper sheets were used for the cathodes.

Research—leaching

Much of the fundamental information about oxidation and leaching of Cyprus ores was developed in laboratory and pilot plant before the main concentrator plant was built. Investigations have been continued, especially in relation to concentrator products for the purpose of developing methods for the recovery of additional copper and pyrites of commercial value. Various methods have been developed for expediting oxidation of the copper to the stage of water solubility, but this is accompanied by oxidation of part of the pyrite, which as a consequence is lost in the solutions, principally as ferrous sulphate. No profitable method has yet been developed for the recovery of this ferrous sulphate (in crystal form, for example), as it has little value

Weathering Operations

All copper minerals in concentrator rougher tailings, except chalcopyrite, can be readily converted to water-soluble chalcanthite under climatic conditions existent in Cyprus for eight months out of each year by submitting such tailings (after they have been partially dewatered by pond settling or filtering) to one of several "weathering" operations. In these the moist material is exposed to slow drying in the presence of entrained air, which supplies the necessary oxygen, and moderate heating from oxidation reactions, under proper conditions. Best conditions are reached when average moisture is about 5 to 6 pct, as this represents conditions when the voids in the mass are partly filled with air, including oxygen, and partly with moisture. Atmospheric humidity is adversely high and temperatures unfavorably low from December to March inclusive. This means that the weathering operation for 12 months' production must be condensed into 8 months. Weathering may be accomplished by any method which pelletizes or breaks up the drained, ponded, or filtered material (normally containing 13 to 14 pct moisture) in such manner as to efficiently entrain air between pellets. This may be done by plowing the surfaces of drained ponds and stripping off this material after it has been "weathered," or by repiling the material in relatively shallow piles—not more than 2 or 3 ft deep —in such manner as to pelletize the material, with ample entrained air between pellets. With best methods it requires only a few weeks to convert all copper minerals except chalcopyrite to water-soluble form. At this stage about 10 pct of the pyrite has become soluble, and will be lost when washing (leaching) the weathered material. Washed tailings will assay about 0.35 pct Cu. The washed tailings will retain slight acidity, in which condition the pyrite may

be very easily floated away from gangue and waste to give a flotation pyrite product assaying about 0.35 pct Cu and 50.0 pct S. This may be separated by classification into granular pyrites and very fine pyrites which may be very suitable for flash roasting.

Some of the chalcopyrite may be converted to water-soluble form by much more prolonged and intensified weathering. Experience shows, however, that about 20 pct of the pyrites will go into solution before the copper content of the remaining pyrites is reduced to 0.20 pct Cu. Optimum overall recoveries of copper from mine ore are consequently dependent on the relative values of copper concentrates and of flotation pyrites. Flotation pyrites with copper content in the range between 0.30 and 0.65 pct is of less value than similar pyrites with less copper or more copper.

Leaching Operations

After weathering flotation tailings and agitating in water the solubles are removed by simple countercurrent displacement of solutions in a series of three or more large thickeners. Clear pregnant copper solutions heavy with ferrous sulphate are removed from the first thickener to go into the copper precipitation plant for precipitation with scrap iron. The first unit in this plant is an acid-resistant tube mill constructed of wood and lined with hard wood blocks with end grain exposed, and partially filled with scrap iron. This is rotated very slowly in order to keep the scrap iron cleaned. Discharged solutions pass to settling vats to separate the precipitate, and discharge solutions from these vats pass to precipitating system, which differs little from the conventional.

Conditions in the first thickener (pregnant solution thickener) are especially unfavorable for settling and clarification of solutions (on account of low pH, etc.). Use of glue and other coagulants has improved this, but settling areas required under these conditions are at least ten times as great as are required for the settling of the same tonnage of copper concentrator tailings. Further intense research on this settling problem is desirable before construction of any larger plant.

The well-washed spigot product from the last of the countercurrent displacement thickeners is pumped to a corrosion-resistant flotation plant where a few rougher cells easily produce a large tonnage of very clean flotation pyrites while rejecting a relatively small tonnage of tailing and a moderate quantity of middlings which goes to cleaners to produce finished flotation pyrites. It should be noted that this pyrite concentrate, which has been floated in slightly acid circuit, is the same as that which was depressed into tailings in the alkaline circuit of the copper concentrator.

We have not yet satisfactorily solved the problem of treating currently produced tailings directly by flotation so as to separate clean pyrites from waste material. Acidulation of the pulp will require large amounts of acid but insufficiently large to give cheap acid production costs. We hope to solve this by using acid ferric sulphate solutions derived from overweathered tailings in stockpiles, and may produce necessary acid and ferric sulphate by aeration of these solutions in combination with SO_2 gas.

Conversion of copper to the water-soluble form by weathering with consequent formation of acid and ferric sulphates requires about 300 to 350 times as long as would be required to do equivalent leaching with the use of hot acid ferric sulphate solutions such as were used for leaching in conjunction with the tests for production of electrolytic copper.

A limited amount of testing has been done to determine practicability of extracting copper by leaching copper *concentrates* after weathering. Advantages to be gained by successful metallurgical treatment along these lines seem nonexistent, however, as the gold-silver values and much of the chalcopyrite are left in the

leached residue and copper precipitates must still be smelted and refined to produce electrolytic copper.

No pyrites products are roasted in Cyprus (for making acid, or for other purposes) and only limited research has been done to determine best methods of treating roaster calcine made from the various flotation pyrites products. Objective is to extract copper and convert the calcine by sintering or nodulizing into suitable and attractive iron blast-furnace feed. During conventional roasting at acid plants much of the copper is converted to ferrites (or ferrates) insoluble in sulphuric acid. Best methods of obtaining good extraction of the copper are not yet developed. The importance of reducing copper to a low figure (e.g. 0.10 to 0.20 pct Cu) is governed by the desires and demands of the iron-ore buyer and the indirect penalties or premiums applicable to the copper content of the sintered iron ore. Low-copper content of leached calcine may be obtained by chloridizing roast preceding leach. In Continental Europe before the war, such practice was well developed on calcines from cupreous fines, but not on calcines from flotation pyrites. A research campaign is now under way. We anticipate little difficulty in sintering the leached flotation calcine to produce a good iron blast-furnace product. This can be done by methods approximating those in use at Copper Hill, Tenn., but calcine production in Europe will be scattered at many roasting plants, and consequently individual sintering operations must be on much smaller scale unless the operators of iron furnaces can be induced to do the sintering, in which case it would probably be better overall practice also to induce them to do the leaching of the calcine. With large concentrated operations, it may also be profitable to extract the gold and silver by leaching with bleaching powder or with cyanide solutions (with regeneration of cyanide) after copper has been extracted.

Research work includes some testing for extraction of copper with ammonia, and also much testing to select best corrosion-resistant alloys, paints, and cementing materials. Extensive research work on "devil's mud" gold-silver ores will not be described in detail.

Ore Dressing and Metallurgy

Skouriotissa Mine Ore

The ore from this mine, containing about 2.25 pct Cu, 48.0 pct S, and 43.0 pct Fe, has always been shipped without concentration to British and Continental sulphuric acid plants. It is prepared in Cyprus by crushing and screening to give: (1) "furnace size" containing lumps under 2½ in. maximum dimension which will not pass through 1-in. square mesh screen, and (2) cupreous fines, consisting of run-of-mine ore crushed to pass through approximately ½-in. square mesh screen.

The cupreous iron oxide calcine or "cinder" residue discharged by acid-plant roasters contains about 3.0 pct Cu and about 60 pct Fe. This is mixed with salt and some unroasted pyrites to supply fuel values and given a chloridizing roast, after which it goes to large vats for a percolation leach. Leach water contains copper, zinc, sodium sulphate, and other chlorides and sulphates. These are recovered by precipitation as cement copper, glauber salts, and so on. The leached residue is covered with bleaching powder and leached again for recovery of a portion of the gold. After this leach the residue is removed from the vats in clamshell power buckets and sintered on Dwight Lloyd type sintering machines to produce iron ore for the blast furnaces. A unique feature of the iron blast furnaces was the "lead well" in which a portion of the residual lead and gold were recovered from the sintered iron ore.

Mavrovouni Mine Ore

The ore containing about 4.2 pct Cu, 48.0 pct S, and 43.0 pct Fe, 0.4 pct Zn,

0.025 oz Au and 0.25 oz Ag has already been described in detail. A typical flowsheet of concentrator operations is shown in Fig 13. Concentration of the copper minerals is difficult and relatively unsatisfactory because of the following conditions:

1. Copper minerals in unoxidized ore include not only the primary mineral chalcopyrite, part of which is intimately associated with the pyrite and finely disseminated, but also bornite and covellite, part of which occurs as microscopically thin films on pyrite, and also "sooty" chalcocite, much of which is less than a few microns in diameter and difficult to float in a pyrite depressing circuit.

Part of the pyrite occurs in very fine crystals a few microns or less in diameter.

Fine grinding will not free the copper minerals from the pyrite.

2. Upon exposure of the ore to the air by development galleries, and after breaking in stopes and galleries, sulphating occurs rapidly. This produces brochantite, a copper sulphate, which is not soluble in water, consequently cannot be extracted and removed from the ore by washing. It is practically inert and does not float well under sulphidizing conditions in alkaline circuit required for depression of the pyrite.

3. Very fine grinding is not economically practical because this reduces the amount of granular pyrites that can be removed as marketable pyrites product from the concentrator circuits.

Methods have been developed and demonstrated by sizeable pilot plants which attain an overall recovery of about 93 pct of the copper by a combination of flotation and leaching, but this is accompanied by a substantial loss of valuable pyrites in solution, and consequently this program of copper recovery is being developed cautiously.

The ore arrives by rail at the crushing-plant storage bins preceding the gyratory crusher. On account of the predilection of the sulphated ores to cement and set, these bins are of the open type without side walls and are discharged through "chinaman chutes," into pan feeders.[8] The pan feeders deliver to the conveyor system, which includes one section with belt 36 in. wide and running at a slow speed, to permit efficient removal of waste, tramp wood, and steel by manual sorters. Crusher is a 10 in. Superior McCulley gyratory. Product of gyratory crusher goes by 24-in. incline conveyor to Hummer screens, from which oversize goes by 24-in. incline conveyor to Allis Chalmers-Anaconda type, 42 by 16 in. rolls, each roll of which is driven by short coupled belt and idler tightener from electric motor. When crushed sufficiently to pass through ½ in. mesh screen, it is conveyed to premill fines storage bin, from which it is drawn regularly at desired rate for concentrator feed (maximum, 90 long tons per hour), and conveyed through weightometer and through sampling section to the premill washing plant. This is equipped with acid proof Dorr classifier, which overflows most of the water-soluble salts and much of the minus 100-mesh copper minerals and pyrites. Wet classifier sands go by incline conveyor to ball-mill feed distributor in grinding section, which contains four 8 ft by 60 in. Hardinge ball mills in circuit with 8 ft by 25 ft 6 in. duplex Dorr classifiers.

Overflow of washing plant classifier is pumped by acid-proof pump to 40-ft diam thickener. Clear overflow from the thickener, containing the soluble copper, is sent to the cement copper-precipitating plant for recovery of the copper. Spigot product from washing plant thickener goes directly to flotation plant or to one of the grinding plant classifiers.

The finished products of the grinding plant go through 50-ft diam thickeners, for regulation of dilution, to the flotation plant which contains 125 flotation cells, equivalent in capacity to 109–24 in. Denver Equipment Fahrenwald cells. Concentrates are cleaned and recleaned to give

copper concentrates assaying approximately 20 pct Cu; 2.0, Zn; 42.0, S; 33.0, Fe; and 0.07 oz Au; 0.70 Ag per long ton. They are pumped to 50-ft diam thickener from which spigot product goes to 11 ft 6 in. by 10 ft diam Oliver filter on which a belt "slapper" has been installed. This slapper consists of a slowly rotating horizontal shaft mounted above the descending side of the drum filter. Attached to this is a strip of rubber belting which "slaps" the filter cake as it rotates, thereby closing the cracks which have formed in the cake. This prevents air leakage through the cracks in the cake and gives a filter cake which is about ½ of 1 pct to ¾ pct lower in moisture than cake produced without the slapper.

Whenever the Oliver filter is out of commission for change of filter cloth or for other reasons, the concentrates are filtered on a 6-ft diam, 6-disc American disc filter. Moisture in Oliver cake is about 12.0 pct. Whenever it is desirable to dry the Oliver cake before storage, it drops directly to the hearth of a 12 ft by 44 ft Lowden drier, which reduces the moisture to about 8.5 pct at a (prewar) cost of about 0.6 s per long ton of concentrates (16 s per long ton of water evaporated). The consumption of fuel oil is 1 lb for 8.5 lb of moisture evaporated. Considerable trouble results from the formation of calcium sulphate as a cake on the drier hearth. Calcium sulphate also builds up in the filter cloth, gradually blinding the cloth. Average service of Oliver filter cloth is about 4500 tons of concentrates. Service of American cloths is about 1100 tons. Oliver filters are wired with stainless steel wire to resist corrosion.

Flotation rougher tailings are pumped at the rate of 1500 to 1700 long tons per day to two Dorr Co. 8-ft duplex classifiers equipped with 16-ft diam bowls. Bowl classifier overflow goes to large hydroseparators. Bowl-classifier sands and hydroseparator-spigot sands are combined to produce a high-grade fine, granular, slime-free pyrites known to the trade as "flotation pyrites," or "pysands." This assays about 49.0 to 50.0 pct S, about 43.0 to 44.0 pct Fe, and about 0.65 to 0.70 pct Cu.

Solids content of overflow from the hydroseparator assays about 2.2 pct Cu, 42.0 pct S, 35.0 pct Fe. It is about 95 pct through 325 mesh. It is stored in ponds which are impounded by dykes built of earth.

Under normal working conditions each 8 ft by 60 in. Hardinge mill had grinding capacity to treat 18 to 20 tons of mill feed per hour. These mills were tested at varying speeds, from 20 to 24 rpm with 20 to 22 tons of ball load. Capacity increased approximately in proportion to power increase, but with a little better power efficiency when mills were operating at 80 to 85 pct of critical speed.

Normal reagent consumption in prewar operations was about the following:

Reagent	Flotation Feed, Pound Per Long Ton
Pine oil	0.265
Amyl xanthate	0.070
Reagent 301	0.070
Sodium Aerofloat Nil to	0.056
CaO	9.3

Chilled white iron ball consumption in grinding was 3.8 lb per long ton of ore.

Treatment of Gold-silver Ores

Metallurgical treatment of the very unique devil's mud gold-silver ores, presented unusual problems. During early mining operations, before wet metallurgical methods had been developed, the selected high-grade crude ore, 1 to 2.5 oz gold per ton, was shipped in cargo vessels to Hamburg for pyrosmelting.

Flowsheet for treatment of the devil's mud gold-silver ores as eventually developed included dry crushing to about ½ in. maximum size by conventional methods. The extremely corrosive crushed ore, although apparently dry, usually contained more than 20 pct of H_2O principally as "water of crystallization," and in addition about 15 to 17 pct of water-soluble materials, including ferrous and ferric sulphate,

calcium sulphate, aluminum sulphate, free sulphuric acid, and silica in solution. The ore was fed at rate of 100 to 150 tons per day to a cylindrical wet-grinding pebble mill constructed of wood and lined with eucalyptus wood blocks set with end of grain exposed to the grinding action. Selected river-bed pebbles of tough, dense diabase (no quartz pebbles were available) were used for grinding media, as iron would not withstand the corrosive action of the ore.

Most of the gold-silver ore disintegrated readily in this acid-resistant plant. The pebble-mill discharge passed to an acid-proof Dorr classifier. Classifier sands, which still contained some corrosive salts, passed into a small Hardinge mill with acid-proof protection between shell and liners. Grinding was done with chilled cast iron balls. Overflow of classifier and Hardinge ball-mill discharge were combined as feed to two standard size acid-resistant Wilfley tables, which produced a good grade of gold-silver concentrates containing pyrite and heavy minerals. Occasionally this table showed a very narrow pencil line of free gold, but this represented a very small percentage of total gold recovery. Table tailings passed over corduroy gold-recovery tables, from which good concentrates were also recovered.

Tailings from these tables were relatively free from sulphides. They constituted the feed to cyanide plant, to which they were delivered through asbestos cement pipelines by duriron acid-proof pumps. First step in cyanide-plant operations was the removal of the corrosive solutions, consisting principally of acidulous ferric and ferrous salts, and gypsum in solution. Approximately 15 to 17 pct of the weight of dry ore was contained in these solutions. Most of the solutions were removed by diluting until approximately 75 to 100 tons of water had been added per ton of dry ore treated, and then dewatering to about 30 pct solids in Dorr type thickener. Apparently this should remove about 95 to 96 pct of the water and of the water-soluble salts. Actually, the thickener spigot product also contained some soluble salts in those particles which had not been ground sufficiently fine to expose the solubles to the solutions.

The thickener spigot product passed to agitating tanks for addition of lime, which averaged nearly 60 lb of lime per dry ton of gold ore. From these agitating tanks it was pumped to the cyanidation agitation tanks. Notwithstanding such careful washing and lime neutralization, consumption averaged from 5.0 to 7.5 lb of sodium cyanide per ton of ore cyanided.

Clear cyanide solutions were obtained from the pulp by countercurrent displacement in Dorr-type thickeners, and by settling the last spigot discharge in the tailings settling ponds. For a time, displacement of pregnant solution was obtained by filtering on American disc filters. The filter cake was very light and "fluffy" as a result of the large amount of limy precipitate in the pulp, and moisture in filtercake could not be reduced below the very high figure of 40 pct.

For several months during one of the dry summers, fresh water was too scarce to supply the large quantities of water required for washing the gold ores. Sea water from Morphou Bay, containing nearly 5 pct of salts in solution, was used for this purpose, without creating any insurmountable difficulties with the exception that approximately 1 oz of gold was lost in the acid ferrous ferric wash water *solutions* per day, as compared with approximately 0.35 oz lost per day in wash-water solutions when using fresh wash water.

These unusual losses of gold *in solution* were determined by evaporating and analyzing the *filtered* overflow of wash waters from washing system. They confirmed our theory that the gold-silver contents of the pyrites had been extracted and concentrated into the devil's mud ore, by meteoric acid ferrous ferric solutions.

Gold and silver were recovered from the cyanide solutions by conventional methods. Precipitate was sold without smelting or refining.

As the material falls on the piles, a considerable portion forms into pellets between which air, with its contained oxygen, is entrained. The tendency of the minerals to oxidize rapidly causes some heating and drying—limited by the amount of oxygen available in the entrained air. Normally this reduces the moisture content from about 11 pct at the filter to about 9 pct at time of shipment.

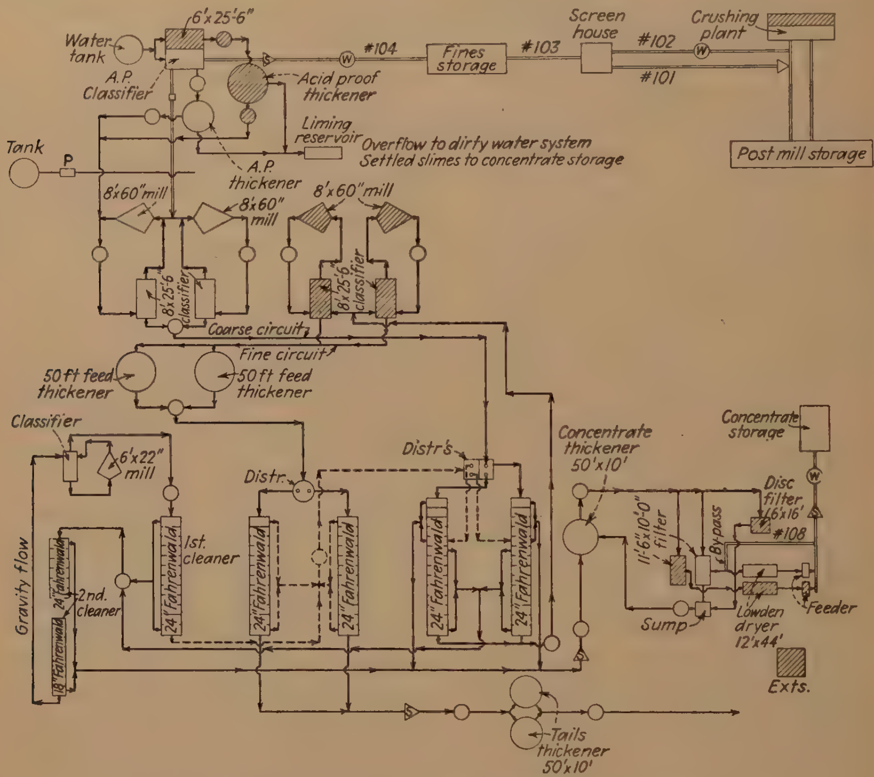

FIG 14—TYPICAL FLOWSHEET OF CYPRUS MINES CORP. CONCENTRATOR.

Wash water from gold ores was carried by ditch to sea, where the iron was precipitated as a reddish brown sludge, which gave a bright tint to the sea water along the shoreline.

STORAGE, LOADING, AND SHIPMENT OF PRODUCTS

Copper Concentrates

The concentrates drop from filters to belt-conveyor system, which culminates in an elevated section about 25 to 30 ft above the concrete floor of the concentrate storage. The concentrates are delivered to this floor by traveling tripper.

Shipments of concentrates are nearly always made in ships-cargo lots of from 3000 to 10,000 tons. Concentrates are loaded from the storage floor, partly by hand and partly by electric-power shovel, into five-ton rocker dump cars hauled by internal-combustion engine tractors, across railway platform weigh scales, to the timber jetty, which extends about 400 ft seaward from the shoreline. (Fig 15). Near the end of this

jetty the concentrates are dumped over chutes to 135-ton-capacity barges placed on either side of the jetty. These barges are towed by either of two company-owned tugs, to the cargo vessels anchored in deep water. There are seven barges. Concentrates are delivered from barges into ships holds by means of ships winches, hoisting one-ton buckets which are loaded with the material in the barges by manual labor. During good weather, operations are continued throughout the 24-hr of the day. Average loading rate is about 2500 long tons per day. Under favorable conditions, rate has exceeded 3500 tons per day. During prewar operations, shipments of concentrates and pyrites productions sometimes exceeded 500,000 long tons per year.

Copper Precipitates

This product results from the precipitation of copper from solutions by means of scrap iron. It usually assays about 60 pct Cu.

FIG 15—LOOKING SOUTH FROM MORPHOU BAY: CYPRUS MINES CORP. JETTY AND BARGE LOADING.

Pyrites

During past operations, pyrites production for shipment has included the following:

1. Cupreous "furnace size"—pyrites lumps ranging in size between 2½ and 1 in. Produced by screening crushed run-of-mine cupreous pyrites ore.

2. Cupreous "fines"—run-of-mine pyrites ore crushed to pass through ½ or ⅝ in. square mesh openings.

3. "Pysands"—a granular slime-free (dust-free) flotation pyrites product containing 0.6 to 0.7 pct Cu and 48 to 50 pct S.

Cupreous "furnace size" and "fines," during normal operations, are put into storage on concrete floors by means of conveyor-belt systems and trippers similar to those mentioned for storage of copper concentrates. The stored material is loaded into five-ton rocker dump cars by means of electrically powered slusher scrapers, which drag the stored material into a hopper mounted on a traveling platform above a depressed track installed alongside the storage piles. The slusher hoist is mounted on the same platform with the hopper, which delivers the material into five-ton cars operating on the depressed track. Loaded cars move material to ships across jetty scales and jetty in the manner described for loading concentrates.

Pysands are stacked in high conical piles on prepared floors. The material is delivered to these piles by pumping through pipe lines to the high point of the stock pile, the sides of the pile being banked as steeply as practical, by manual labor. Alternatively, pysands have been partly dewatered and then carried by conveyors to the high point of the stock pile. The objective in piling the sands as steeply as possi-

FIG 16—LOOKING NORTH: MORPHOU BAY IN BACKGROUND, WITH CARGO SHIPS AT ANCHOR SHOPS, AND TAILINGS PONDS, SHOWN IN CENTER

ble is to get better drainage, in order to reduce moisture to about 2½ to 4 pct for shipment.

Material from pysands stock pile is loaded by manual labor or by power shovel into five-ton rocker dump cars which pass over weigh scales and jetty into barges for transfer to ship holds.

General

Central power plant[9] consists of six Crosley Premier 8-cylinder diesel engines, supercharged to rated capacity of 1440 hp each, and connected to British generators operating at 2200 v, 3 phase, 50 cycle. Power is transmitted at 2200 and 11,000 v. A few motors operate at 2200 v, most at 440. Synchronous motors and static concensers are used to maintain power factor.

Fuel oil is purchased as "fuel oil" and as "gas oil," the latter being lighter and more volatile. The fuel oil is used for firing steam boilers and similar purposes. For diesel engines we usually mix the two, using from 40 to 50 pct of the more expensive gas oil. Oil is received in Motorvessel tankers and pumped through submarine oil line to several oil tanks, which have total capacity of about 7000 tons of oil, (50,000 bbl). Gas oil and fuel oil are stored separately and also as a mixture. Oils are distributed to power plant and other points of usage by pumping through steel pipelines. Design of submarine pipeline and method of laying in place are of special interest. Pipe is 6 in steel line coupled with Dresser screwed sleeves, which were caulked with copper wire after coupling. Steel pipe and sleeves are coated with hot tar and wrapped with Hessian cloth or similar fabric This is again coated with hot tar and fabric and a third coat of tar. This is enclosed in wood staves bound into the form of wood pipe by means of copper wire. Over this is another protective wrapping. To the submerged end of the pipeline is attached the flexible portion of the pipeline, consisting of three or four 25-ft lengths of steel reinforced rubber suction and pressure hose with oil-resistant rubber lining. Hose is wrapped with Manila rope for additional protection. Flexible end of hose is weighted and anchored to sea bed, when not in use. It is marked by buoy, which is attached to the free end of hose by chain which is used for bringing free end to surface when receiving oil from tankers.

During construction of submarine line, the pipe was supported on four-wheel trucks mounted on 30 in. gauge track, which extended inland on a fairly regular grade of about 2½ pct, normally to the shore line. Entire line was launched and submerged in 27 min. during calm weather by supporting the line as launched by means of floats consisting of empty 5-gal gasoline tins arranged in pairs. When line was all afloat, these tins were punctured by

XERO PLANT OF CYPRUS MINES CORP. INCLUDING CONCENTRATOR, CYANIDE PLANT, POWER-PLANT, OF PICTURE. VOUNI HILL IN LEFT BACKGROUND.

men in rowboats, allowing the line to sink to sea bed. That portion of the line which would be subject to wear near shore line was protected by an overlying blanket of heavy steel rails, attached together in parallel, and lying parallel to pipe.

Mine compressed-air system is served by compressors driven by direct connected diesel engines. This makes some saving in power installation and operating cost, as compared with electric-driven compressors using power from central power plant.

Water system serving Xero and Mavrovouni areas includes Byron-Jackson deep-well pumps in churn-drill holes, which tap submarine fresh water reservoirs in sand and gravel beds which dip seaward under more impervious strata. These are fed during the wet seasons through river beds which have been depressed by comparatively recent geological subsidence of that section of the Island. Deep-well pumps deliver to one large concrete reservoir set on hill slope at about 100-ft elevation, and to several smaller reservoirs. Water for the Mavrovouni mine area is pumped about four miles by triplex pump through 4 in. line to tanks set at about 400 and 600-ft elevation. From these it is piped for surface and underground use.

Principal shops are located at Xero (Fig 16). These are well equipped, and include electric shop equipped for almost all kinds of motor, transformer, generator, and transmission-line repairs; machine shop with large lathes and presses suitable for repairs to concentrator machinery, locomotives, diesel engines, compressors, hoists, etc.; blacksmith shop with heavy forging hammers, boiler-plate and tin-plate shops and electric welding machines; locomotive and railway car repair shop; carpenter shop well equipped with woodworking machinery, and suitable for manufacture of windows, doors, household furniture, patterns, etc. There is also a foundry and pattern storage building. Foundry is equipped for making gray iron, white iron, and chilled castings and brass and bronze castings. It has a Plessman ball-casting machine in which are made all the balls for concentrator grinding. The foundry produces practically the entire requirements of grinding balls, ball-mill liners, centrifugal pump bowls, runners, and other wearing parts. Foundry and machine shop are frequently called upon to manufacture replacement parts that cannot be delivered soon enough. Pattern shop contains several thousand patterns.

The Company's Medical Department has a large, well-equipped hospital with 50 beds, dispensary, operating theater, X-ray room and equipment, autoclaves, office quarters, kitchen, nurses' living quarters, laundry, and residential quarters for staff consisting of Chief Medical Officer and family, two assistant medical officers, dis-

penser, matron, nurses, etc. Sanitary Division of the Medical Department includes antimalarial crew and several sanitary inspectors whose territory covers villages, change houses and surface and underground latrines and sanitation in general. Daily clinics are held at hospital and at two or three other central points.

Social Service department works in close cooperation with Medical Department. It is staffed by three or four supervisors and several assistants. It supervises and arranges athletic and social events and club activities for children and youth of both sexes. It distributes milk from Company's homogenizing plant to school children. Educational classes are taught gardening and several homecrafts at two or more welfare centers. A day nursery is conducted.

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Use of Jumbo Drilling Machines in the Tri-State District

By S. S. Clarke,* Member AIME

(New York Meeting, March 1947)

Late in 1942, the increasing demand for zinc, coupled with the growing shortage of miners and the knowledge that some abandoned mines would have to be reopened for prospecting and development, led to considerable thought as to the possibilities of further mechanization in order to conserve man power.

The ore bodies, locally termed "sheet ground," are an important ore bed in some mines and are in the lower part of Boone formation of the Mississippian Series. Some mines contain interbedded layers of nearly pure galena and sphalerite in horizontal sheets ranging in thickness from a thin seam to several inches and is persistent over large areas. This type of ore body offered the best opportunity for study and experiments. The use of Jumbo drill carriers was advanced but some doubt was expressed as to the adaptability of Jumbos as then in use to mine headings as compared to post and arm mounting (Fig 1). Another consideration was the introduction of truck haulage when several mines were trackless.

A drill frame, built to carry 2 4-in. drifters, was mounted on a truck chassis. This design proved a failure as it was not rigid and the resiliency of the tires and springs allowed too much play. Long screw jacks were then used to hold the truck firmly in place, but placing the jacks and securing the truck required more effort and time than setting up post drills, so the truck carrier was abandoned.

The use of a bull dozer underground had been under consideration for some time to be used in "brunoing" the broken dirt from behind pillars and other inaccessable places so the slusher drags could load without double dragging the dirt. A dozer with hydraulic blade lift was purchased and R. I. Tuthill, Superintendent of Section 30 Mines, suggested that a long drill arm be substituted for the blade, and the drills mounted on the cross member. The hydraulic lift would permit drilling holes at any required spacing. The "cat" was very mobile and answered the problem of a trackless mine. The initial tests showed that the "cat" would make a good carrier, but the hydraulic controls were not positive enough to keep the drills in alignment while drilling. The next step was to build a small hoist. This hoist was attached to the power take-off on the "cat" and by means of sheaves and a double line of ⅜-in. hoist cable, a positive control for raising and lowering the boom was obtained. To avoid serious injuries to the drillers should the cable break, a safety chain was fastened to the drill boom and hooked into a slot in the back of the "cat" frame.

A different power unit was obtained for the "cat," as a state law prohibited the use of internal combustion engines in mines. A custom machine shop had just placed on the market a small hoist powered by a Model *A* Ford engine that had been very ingeniously converted to a slide valve engine using compressed air. To use this unit on a "cat," our master mechanic de-

Manuscript received at the office of the Institute March 24, 1947. Issued as TP 2294 in Mining Technology, March 1948.

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vised a very simple reversing device. While the efficiency of this type of engine was low, the moving time of the Jumbo during a shift was slow and efficiency was of no consequence. The engine developed about 15 hp which was ample to propel the "cat."

FIG 1—LOTTSON NO. 2 DRILL ON POST AND ARM.

Several Jumbos were assembled on caterpillar No. 30 powered with either a converted "Model *A*" motor or converted "cat" motor (Fig 2). As deliveries on caterpillars became very slow, for uniformity we began to assemble Jumbo drilling booms on the Eagle-Picher crawler which had been developed and made in our shops to carry the slusher hoists and ramps. These crawlers had a lower, wider tread, and a lower center of gravity (Fig 3). The possibility of turning over was eliminated. One nearly fatal accident occurred in the early stages of development when a Jumbo, in backing out of position with the boom nearly vertical, backed into a small sump which caused the rear end to drop about 2 ft causing the boom to swing past center and pin the operator to the seat. Repetition of similar accidents was prevented by welding bumper straps to the frame. This prevented the boom from passing the vertical.

The E-P crawlers are powered by either two 5 hp gear-head electric motors or two air motors, piston type. Traveling speed is about 100 fpm. Both power units are satisfactory, but the air motor drive eliminates stringing electric current to the Jumbo.

The first drill boom was rectangular in shape built of 3½-in. od pipe, 10 ft long and 6 ft 6 in. wide with an *A* truss to strengthen the longitudinal members and form an anchor post for the sheave carrying the hoisting cable. As this type of boom required the drillers to be inside the frame while drilling, this design had a definite hazard should the cable and safety chain fail. A *T* head boom was then designed.

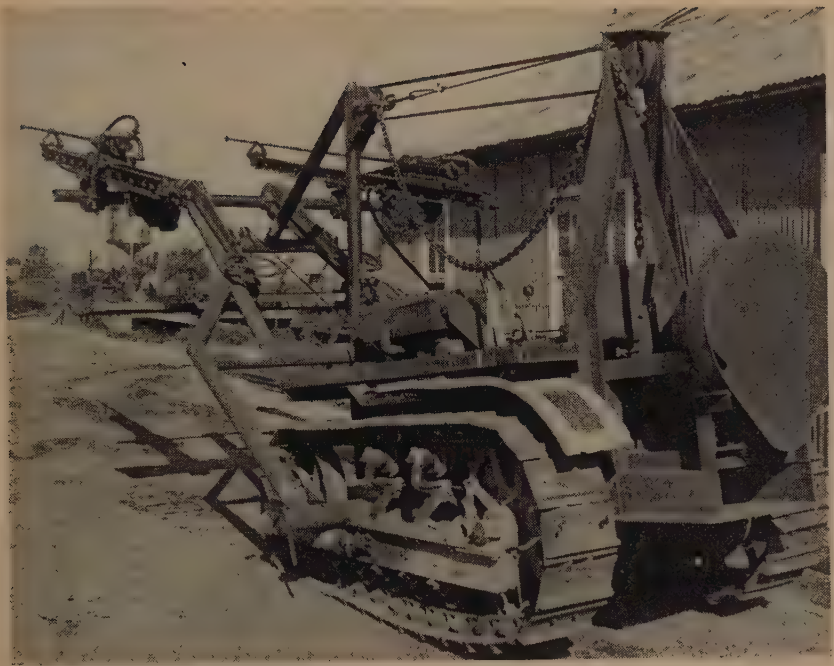

FIG 2—JUMBO DRILLING BOOM ASSEMBLED ON CATERPILLAR NO. 30.

FIG 3—JUMBO DRILLING BOOM ASSEMBLED ON EAGLE-PICHER CRAWLER.

This permits the operators to be in the clear while working and is the standard type now in use.

to the drifters. Two large drill oilers are also mounted on the Jumbo.

Drifters were first mounted on 36-in.

FIG 4—DRILL SASHES ATTACHED TO JUMBO.

FIG 5—JUMBO DRILL CARRIERS POWERED BY AIR MOTOR AND TRAVELING ON "CAT" TREADS.

automatic sashes. These were so satisfactory that a longer sash was thought of. One long enough to take a 10-ft steel which was

The boom also acts as a small air receiver with moisture drains. The bottom pipe strut is used as manifold for the water lines

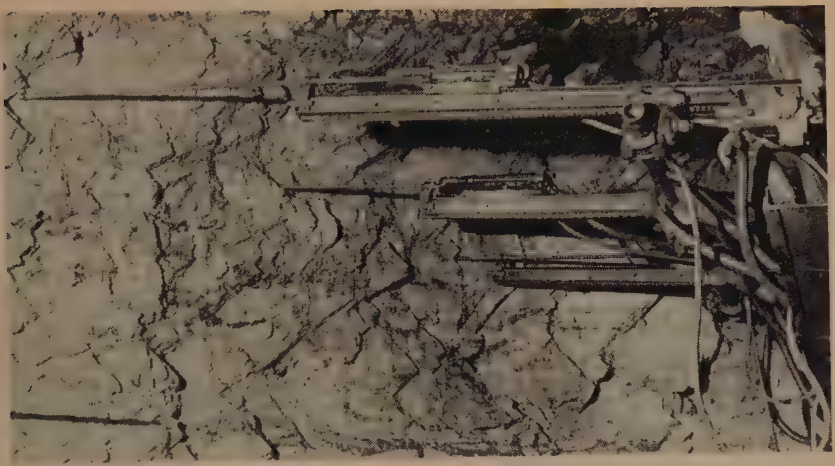

FIG 6—THE ADAPTATION OF 10-FT FEED SASHES PERMITS COLLARING AND BOTTOMING A HOLE WITH ONE LENGTH OF STEEL.

FIG 7—HEADING-HOLES LOADED.

the average depth of breast holes, thus one steel could start and finish a hole with a minimum amount of effort on the part of the driller and a reduced stock of drill steel. A sash 11-ft long was built using the conventional screw feed as a test. The results were encouraging and long sashes were then built, patterned after the typical wagon drill sash only much lighter in weight as a heavy slab-back was not necessary for horizontal drilling. A small reversible turbine air motor propelled the drill by a chain and sprocket drive. The success of this arrangement was immediate and it reduced the work load to such an extent that one experienced miner and a helper could handle two or three drills, thus permitting one skilled miner to each Jumbo (Fig 4–6).

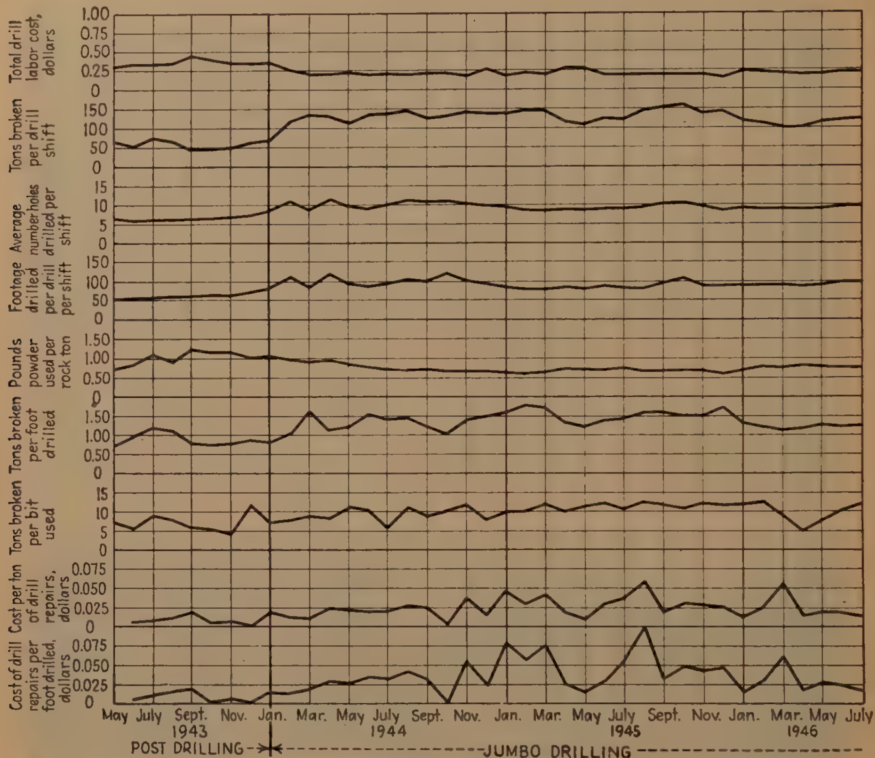

Fig 8—Graph showing difference between post and Jumbo drilling.

Drilling patterns were standardized as much as possible and the heading could be kept in good shape to permit maximum results from a round. The footage per drill was doubled and by uniformity in proper spacing of blast holes the tonnage per drill was more than doubled.

One question arose as to the increased steel breakage, the theory being advanced that as one piece of steel would do the work that was previously divided between 5 pieces, breakage would be excessive. Several years of experience have shown that such is not the case. The steel consumption is arrived at by actual weighing of the entire stock of drill steel each six months which gives us an accurate figure (Table 1).

The miners at the West Side Foley mine which is in "M" bed[1] and is about 40-ft

[1] G. M. Fowler and J. P. Lyden: The Ore Deposits of the Tri-State District (Missouri-Kansas-Oklahoma). *Trans.* AIME (1932) **102**, 206.

thick asked for some Jumbo machines. After some hesitancy, one was assigned to them. The ore run was of sufficient width that two or three days drilling could be done while the slusher drags were cleaning up the other heading.

TABLE 1—*Data on Steel Breakage*

Year	Mounting	Pieces of Steel Broken	Breaks per Drill Shift	Feet Drilled per Break	Lb of Steel Broken Rock Ton
1943	Post	1,532	0.61	101.0	0.0622
1944	Jumbo short sash	1,503	0.71	142.9	0.0672
1945	Jumbo long sash	1,541	0.86	104.2	0.0861
1946	Jumbo long sash	1,840	0.69	128.3	0.0760

After a few days spent in shaping up the heading, the underhand benches or stopes, as locally named, were drilled and blasted. The angle of repose of the broken ore rock permitted the Jumbo to climb the pile and drill out the heading rounds. We believe that a 30-ft climb is about the economic limit for moving up, as any higher climb reduces the time available for drilling. In ground 40 ft or higher we drill the heading rounds from a post or column mounted drill, using the Jumbo for bench drilling.

The drilling pattern in sheet ground is now standard (Fig 7). The irregular drilling breaks which formerly affected the breast contour caused by lack of cooperation between drillers, poor judgment, and failure of one drill to complete a round, were factors that frequently resulted in "duck nests" and necessitated short rounds to shape the ground. With Jumbos the breaks are uniform as a full double round can be drilled from one setting and the use of tripods for drilling the stope holes has been eliminated.

The drill labor costs have been lowered from $0.38 per rock ton to $0.24 per ton, this in spite of increasing wages as listed below:

	PER 8 HR SHIFT
Jan. 1, 1943 to Aug., 1943	$6.55
Aug., 1943 to Apr., 1946	7.05
Apr., 1946 to Dec., 1946	8.05
Dec., 1946 to present	9.03

In addition to base pay, Jumbo crews are paid a bonus of 8¢ per ton for all tonnage above base which on 12 to 14-ft ground is 110 tons. Bonus pay is included in the cost.

Fig 8 graphically shows the difference between post and Jumbo drilling. Table 2 gives a recapitulation of the graph.

TABLE 2—*Operating Statistics*

	Post Mounting	Jumbo Mounting
Drill labor per rock ton	$ 0.38	$ 0.24
Tons broken per drill shift	58.0	130.0
Holes drilled per drill shift	6.4	9.7
Footage drilled per drill shift	60.4	95.6
Tons broken per bit used	6.5	10.0
Lb of powder per ton broken	1.02	0.75

Jumbo construction cost ranges from $3,000.00 to $5,000.00 depending upon the type of crawler and power units used. This does not include the cost of drifters, and the weight will range from 10,000 to 11,900 lb.

The Missouri School of Mines has published a detailed bulletin on Jumbo drilling by Forrester and Taylor.

Observations of the Relation of Drilling Speed to the Size of Cuttings*

By Tell Ertl† and Ernest E. Burgh,† Members AIME

(New York Meeting, February 1948)

Introduction

The Bureau of Mines is operating an oil-shale mine 10 miles west of Rifle, Colo., as part of its Synthetic Liquid Fuels program. The purpose of operating this mine is twofold: First, to supply oil shale to an experimental processing plant; and second, to develop practices and select methods for mining oil shale on a commercial scale at the lowest practicable cost.

The second purpose—that of finding out how to mine oil shale most cheaply—entails considerable mining research. Since the drilling of oil shale probably will be one of the major items of the mining cost, factors that influence drilling speeds were investigated. One phase of the drilling program was the development of a successful hard-surfaced rock-drill bit[1] that can be used for drilling long holes in oil shale without changing bits. During that phase, drilling speeds were measured and drill cuttings were collected for assay and for screen sizing. When the cuttings from the individual tests were compared, a definite relationship was shown to exist between the drilling speed and the coarseness of the drill cuttings.

Procedure

Measurement of Drilling Speeds

All drilling in the tests was done with a Model UMB-D-99W Gardner-Denver wagon drill. Round-lugged drill steel, 1¼ in. in diameter was used in 6, 12, and 18 ft lengths with 2⅛-in. Timken H-type bits hard-faced with acetylene tube borium. The tests were made in the lower adit of the Bureau of Mines oil-shale mine; the grade of the oil shale drilled ranged from 18.6 to 42.7 gal per ton. During the experiments air pressure varied between 75 and 90 psi while water pressure remained fairly constant at 75 psi. All holes were drilled almost horizontally.

Each drill hole was thoroughly cleaned of cuttings after being collared. The advance of the drill along the carriage was measured and recorded as the drilling speed in inches per minute.

Method of Collecting Cuttings

To facilitate the collection of cuttings, each drill hole was collared in a smooth, vertical portion of the oil-shale breast (Fig 1). A galvanized-steel sheet was fastened to the breast below the area in which the hole was to be drilled. The cuttings washed out of the drill hole by the drilling water were guided by the steel sheet into a wooden box. The cuttings that remained in the drill hole after the drill was withdrawn were scraped out and placed in the same box as the cuttings washed out of the hole.

After the cuttings from a 1-min. drilling interval were collected, they were placed in flat metal pans. The fines were allowed to settle and the clear water was decanted. The cuttings then were dried in the sun.

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* Published by permission of the Director of the Bureau of Mines. Manuscript received at the office of the Institute Dec. 10, 1947. Issued as TP 2409 in Mining Technology, July 1948.

† Mining Engineer, U. S. Bureau of Mines, Rifle, Colo.

[1] References are at the end of the paper.

Determination of Surface Area of Cuttings

The dry cuttings were split in a Jones Riffle sampler; one portion was sent for assay and the other retained for screen sizing. Screen sizing was done with a Ro-tap testing sieve shaker. Standard Tyler sieves of 3, 4, 6, 8, 10, 14, 20, 28, 35, 48, 65, 100, 150, and 200-mesh sizes were used. The material retained on each sieve was weighed, and the percentage of the total sample retained on each sieve was calculated.

FIG 1—COLLECTING DRILL-CUTTING SAMPLES IN OIL-SHALE BREAST.
Photograph by U.S. Bureau of Mines.

A more accurate method of comparing the coarseness of the cuttings than a screen-size tabulation was sought, and it was decided that the surface area of the cuttings could be calculated. The area of the plus 200-mesh cuttings was calculated readily for each of the samples collected, but the area of the minus 200-mesh cuttings was not obtained. The minus 200-mesh cuttings were disregarded, as the surface area of those particles probably cannot be obtained with any degree of accuracy.[2] Moreover, some of the minus 200-mesh particles were lost in collecting the samples and during screen-sizing operations.

After the samples had been sized, a microscopic study of their shape was made. It was discovered that the particles could be considered to be rectangular parallelepipeds with sides that varied in length as x, $2x$, and $x/4$, where x is the average screen aperture. The average screen aperture is the arithmetic mean of the size opening of the screen through which the particle passed and the size opening of the sieve on which the particle was retained.

The average volume of each particle then is found to be $\frac{X^3}{2}$. The average particle weight, in grams, was calculated by multiplying the volume, in cubic centimeters, by the specific gravity.

The average surface area of each particle was found to be $5.5X^2$. Since the average surface area of each particle and the average particle weight are known, the surface area per gram of sample retained on

each screen is easily determined. By multiplying the grams retained on each screen size by the surface area per gram for that screen size, the total surface area can be drilling speed the smaller is the surface area of the plus 200-mesh drill cuttings.

The second function also was graphed against the drilling speed (Fig 3). That

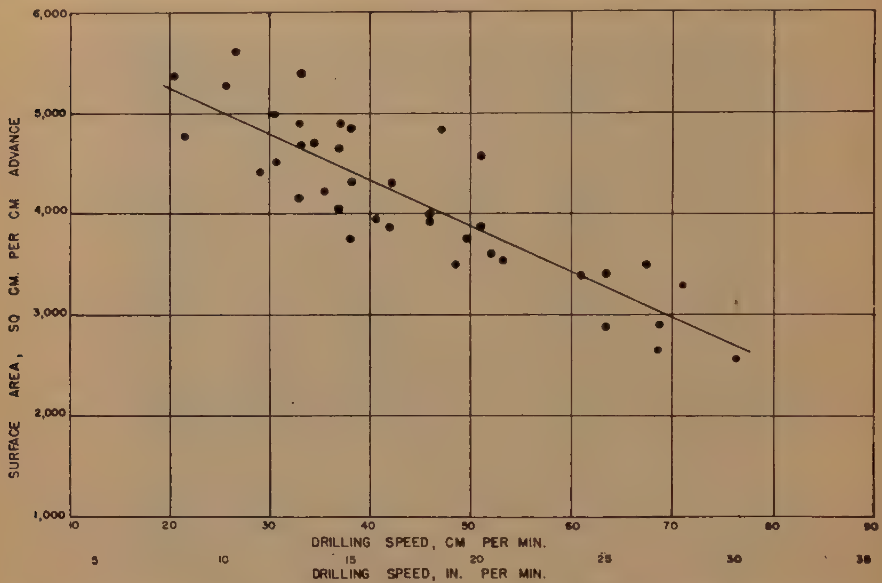

Fig 2—Relation of surface area of rock-drill cuttings (plus 200-mesh only) per centimeter advance to distance drilled per minute.

computed. The percentage retained on each screen is used as the grams retained, so that the total surface area computed for each sample can be compared on the basis of a 100-g sample.

Relation of Drilling Speeds to Surface Area of Cuttings

When the surface areas of the plus 200-mesh drill cuttings were compared with the corresponding drilling speeds, a definite relationship was noted. For the purpose of graphing, two functions were calculated for 39 samples: (1) Surface area formed per centimeter advance of the drill hole; and (2) surface area formed per minute of drilling. The first function was graphed against the drilling speed (Fig 2). The graph is a straight line: The faster the

figure shows that the rock drill used apparently could not produce more than 250,000 sq cm of surface area per minute of drilling in oil shale. This indicates that the rock drill probably could not drill in oil shale at a speed much greater than 80 cm or 32 in. per minute.

Conclusion

The data presented show conclusively that the speed with which oil shale can be drilled with any one percussion drill varies directly with the size of cuttings produced. This statement substantiates the assumption of Forbes and Barton who, after a few tests, concluded that the speed of percussion drilling of rock "seems to be proportional to the coarseness of the cuttings."[3]

Furthermore, the same data indicate

(Fig 3) that there is a definite limit to the surface area that can be produced by any one rock drill during a given period.

The value of the conclusions to the mining industry is as follows: The first conclusion—that the drilling speed varies directly with the coarseness of the cuttings —indicates that, for efficient drilling, bits should be designed to produce coarse cuttings and *that* cuttings should be removed from in front of the bit as quickly as possible.

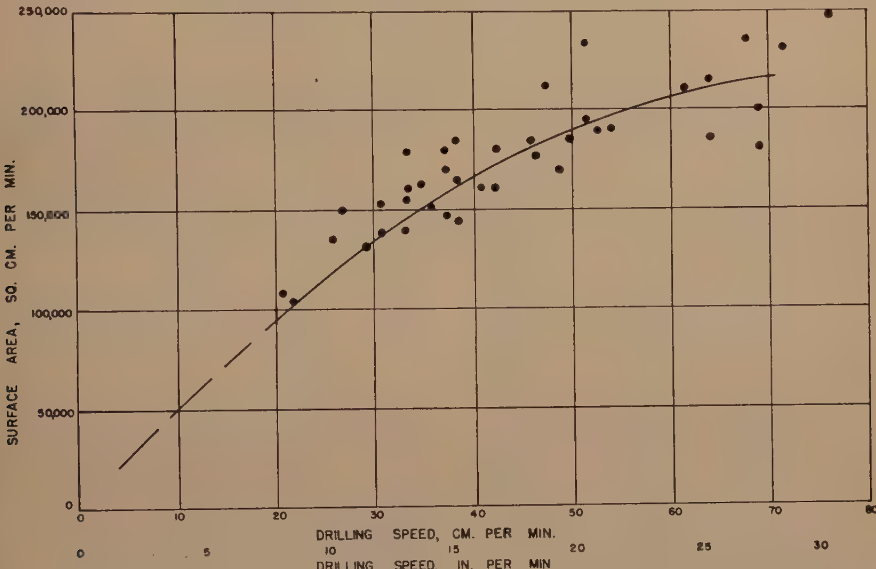

Fig 3—Relation of surface area of rock-drill cuttings (plus 200-mesh only) per minute of drilling to drilling speed.

The second conclusion—that each drill is limited in ability to produce new surface area and consequently in drilling speed—also can be useful to the mining industry. Many investigators[4] have shown that the production of new surface area is directly proportional to the energy consumed in producing that surface area. Any drill is limited in the ability to produce new surface area because it is limited in its energy output. Therefore, it appears that increasing the energy output of a drilling machine is a factor in increasing the speed of drilling.

Summary

The speed of percussion drilling varies directly with the coarseness of the cuttings produced and speed of drilling is limited by the energy output of the drill. Rock-drilling speeds could be increased, (1) by improving the design of rock bits to produce coarser cuttings and to permit a more ready removal of the coarser cuttings, (2) by removing the coarse cuttings as rapidly as produced, and (3) by using rock drills with greater energy output.

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4. Pp. 135–136 of ref. 2.

Studies of the Design of Shaped Explosive Charges and Their Effect in Breaking Concrete Blocks

By George B. Clark,* Member AIME

(New York Meeting, March 1947)

The "Munroe effect" of shaped explosive charges was discovered by Charles E. Munroe more than 50 years ago (in 1888), but it was not until World War II that it was put to any practical use. Both Allied and Axis armies used the principle effectively in bazooka projectiles, artillery projectiles and in fixed shaped charges. The first two were used primarily to combat tanks and other armor and the latter to assault pillboxes and other fixed concrete emplacements.

Many engineers and others who saw the military application of the jet effect of shaped charges believed application would be possible in many industrial fields where blasting is used, such as mining and quarrying. Some work has already been done on the problem of making shaped charges applicable to mining operations. Experimentation conducted at the National Tunnel and Mines[1] was directed primarily along the lines of use of shaped charges for secondary blasting in connection with long-hole stoping methods.

Experimentation initiated earlier and conducted by the author[2] at the University of Utah on a more scientific basis was directed at revealing the basic principles of the functioning of shaped charges, as well as the feasibility of using them in mining operations after the principles of their operation had been clearly defined. It was found that: (1) a given design of charge required an optimum standoff to make it most effective, either in penetrating solid rock or steel (see Fig 3); (2) for a given design of charge there exists an optimum ratio of charge diameter to charge height above which the increase in effectiveness of the charge decreases rapidly (see Figs 4 and 5); (3) partial confinement has the effect of making charge performance much more consistent; (4) the higher power and higher velocity explosives tested are much more effective than relatively low-strength explosives; and (5) shaped charges may be adaptable to mining operations in secondary breakage and drilling blasting holes.

Fig 1 is a diagrammatic sketch of the action of a shaped charge for two types of cavities, while Fig 2 shows the approximate manner in which the jets are believed to be formed when an explosive wave encounters the cavity, approaching it from the top.

The experimentation already completed has laid the foundation for further research in the study of shaped charges. Standoff effects, effect of the strength of explosives and other features have been fairly well established. Such elements in the design of shaped charges as (1) geometry of cavity liners, (2) relative horizontal dimensions of explosive column, (3) effect of various metals and alloys in cavity liners, (4) percentage studies of jet energies, and

Manuscript received at the office of the Institute Dec. 16, 1946. Issued as TP 2157 in Mining Technology, May 1947.

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[1] References are at the end of the paper.

(5) effect of complete confinement, as well as other features of design, remained to be investigated for possible industrial applications.

The research described herein was directed along some of these lines of investigation to attempt to fix the value of further studies in the same line as well as to establish methods of investigation that would yield satisfactory results. Complete answers to all of the problems involved will be revealed only after many more months of intensive research. The results described herein, however, represent a marked step forward in the direction of a complete understanding of the functioning and possible industrial use of shaped explosive charges.

Results of Tests

Wall Thickness

Three-inch cones (Fig 6) were designed to fit into 4-in. pipes to obtain the advantage of width of explosive around the base of the cone, if any such advantage exists. Groups of cones varying in wall thickness from 1/16 to 3/16 in. in steps of 1/32 in. were inserted into charges and fired against granodiorite as a target. The results are shown in Table 1. Penetration values for each group were averaged and the results plotted in Fig 7.

It was assumed at the beginning of the present research that wall thickness of liners should vary only with size of charge; that is, the wall thickness should be increased as the diameter of the charge. Results indicate that apex angle of the cone also affects the optimum wall thickness of cavity liners. In previous experiments penetrations up to 20 in. were obtained with the same explosive charge and with cast-iron cones 3/32 in. thick and apex angle of 45°. With 60° cones of the same thickness, however, penetrations of from 6 to 10 in. were consistently obtained in the present tests. Fig 7 indicates that the optimum wall thickness for a 60° 3-in. diameter cast-iron cone is 3/16 in., or just double the thickness assumed to be optimum from tests with 45° cones.[2]

The findings here only serve to emphasize the fact that in the design of shaped charges the various factors that affect the performance of the charge are interdependent.

Tapered Cone Walls

Table 2 shows the effect of different wall tapers upon the penetration effect of cast-iron jets in granodiorite. The best results were obtained in group two with a wall thickness of 2/32 to 5/32 in. from apex to base. If, however, the lowest results in each group are ignored, there is not an appreciable difference in the

Table 1—*Effect of Wall Thickness of Uniform Wall Cones on Penetration in Granodiorite*

3-inch 60° Cast-iron Cone; 100 Per Cent Blasting Gelatin

Shot No.	Wall Thickness, In.	Spall, In.	Depth of Hole, In.	Total Depth, In.	Diameter of Hole, In.
1	2/32	2	2	4.	1+
2		3	2	5	
3		2	2	4	
				Av. 4 1/3	
1	3/32	3 1/2	4	7 1/2	1
2		2	3	5	1
				Av. 6 1/4	
1	4/32	3	7	10	1 3/4
2		4	4	8	1 1/4
3		3	8	11	1 1/4
				Av. 9.7	
1	5/32	3	8	10 1/2	1 1/2
2		3	10	13	1 1/2
3		2	7 1/2	9 1/2	1 1/2
				Av. 11	
1	6/32	3	8 1/2	11 1/2	1 1/2
2		3	8 1/2	11 1/2	1 1/2
3		3 1/2	5 1/2	9	1 1/2
				Av. 10 2/3	

results obtained from all five groups tested. A comparison of the results for uniform wall cones (Table 1) and tapered wall cones (Table 2) shows that tapered

Effect of Confinement

Cast molybdenum-steel cases of dimensions shown in Fig 8 were used. Uniform-wall cast-iron cones, 60° 3-in. diameter,

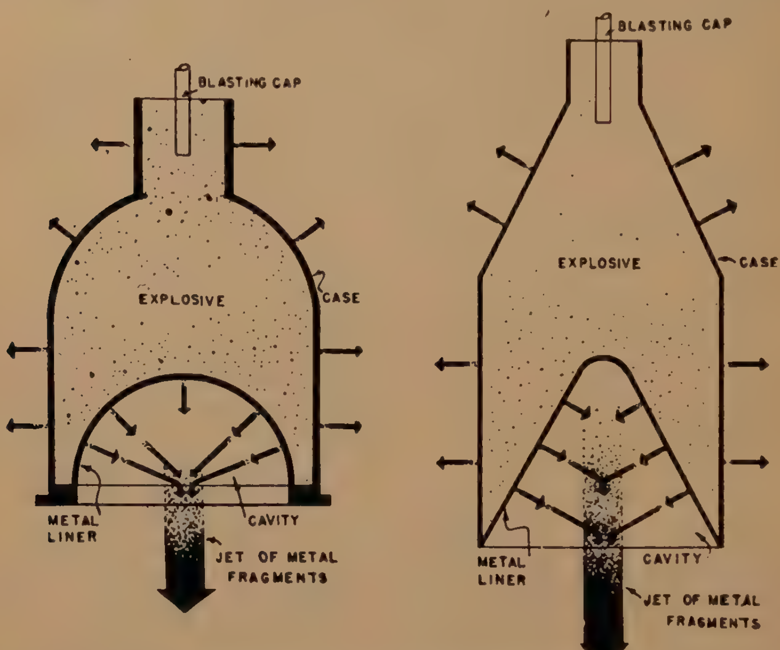

FIG 1—SKETCHES OF SHAPED CHARGES WITH (*a*) HEMISPHERICAL AND (*b*) CONICAL CAVITIES SHOWING MECHANISM OF FORMATION OF MONROE JET.[2]

wall cones give consistently higher values of penetration. A theoretical reason for this improved performance is discussed under the theory of jet formation.[4]

The increase of efficiency of tapered cones might also be explained on the following basis: For small cones the optimum wall thickness is relatively small for uniform wall cones. For larger cones of the same apex angle the optimum wall thickness is proportional to the size; i.e., base diameter of the cone. Consequently, a cone that has a wall thickness proportional to the distance of a point in the wall from the apex of the cone may provide optimum conditions for collapse at any horizontal section of the cone.[4]

$\frac{3}{32}$-in., were used as liners. Eight shots were fired on granodiorite; the tabulated results are given in Table 3. With increased thickness of case, which in effect formed a sort of breechblock for the charge, the jet being the projectile, there was not such a marked increase in depth of penetration as was expected, even though the increase was considerable. The increase in diameter and volume of hole is noteworthy and can be laid directly to the effect of the confinement of the explosive charge.

The high strength of the 100 per cent blasting gelatin was more than sufficient to shatter the large 3-in. case into small fragments. Only one small fragment from this case could be found in the vicinity

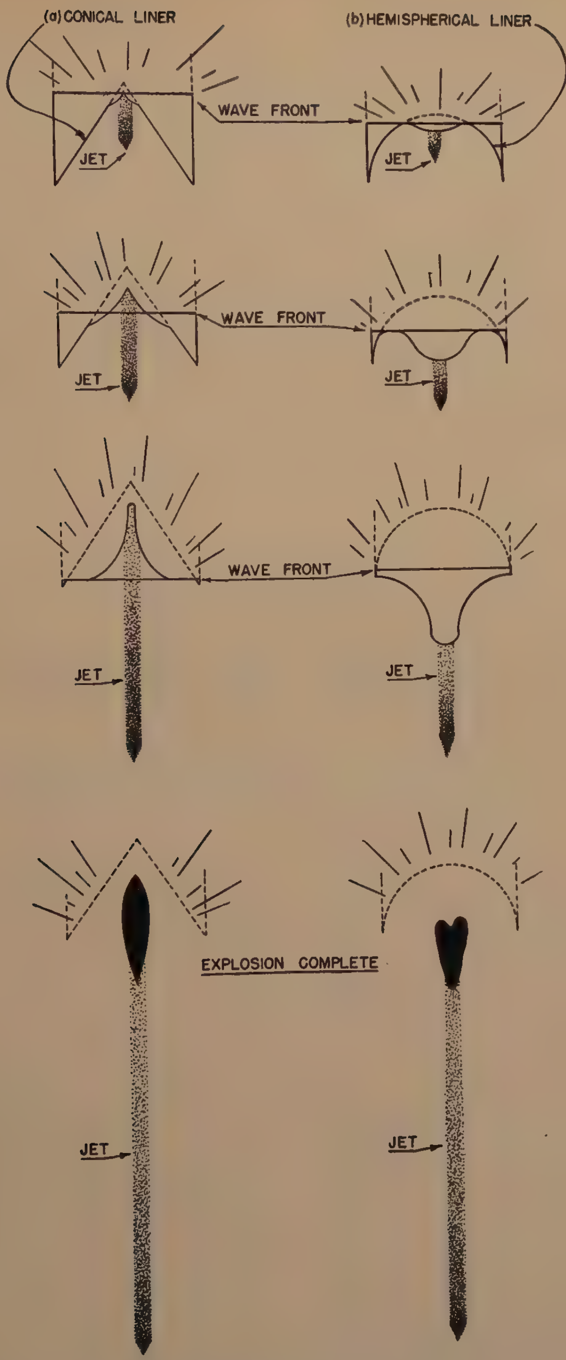

FIG 2—THEORETICAL DIAGRAMMATIC SCHEME OF STAGES OF COLLAPSE OF (*a*) CONICAL AND (*b*) HEMISPHERICAL CAVITY LINES.

where the charge was fired. The thickness of steel case that would be necessary to hold such a charge without fracture is not known. Breechblocks of artillery pieces are of comparable dimensions (3-in. thickness), but the strength of projectile propellents is only a fraction of that of 100 per cent blasting gelatin.

The total effect of confinement in these tests upon the performance of shaped charges was to increase both depth and diameter of the hole. The mechanics

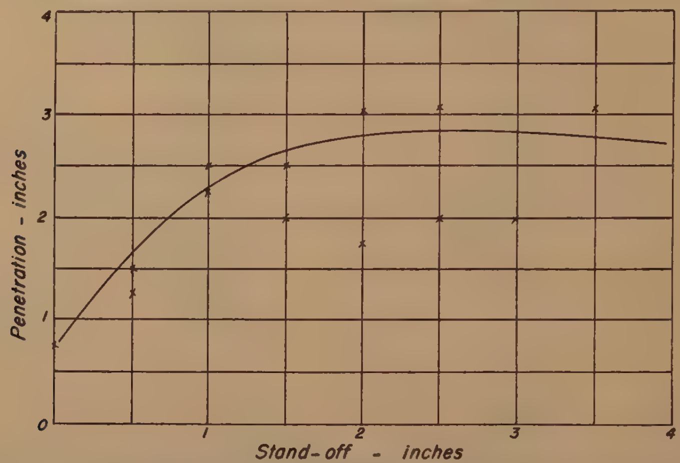

FIG 3—EFFECT OF CHARGES WITH 1¾-INCH DIAMETER HEMISPHERICAL LINERS OF CAST IRON ON STEEL PLATES WITH VARIABLE STANDOFF.[2]

TABLE 2—*Effect of Tapering Walls of Cones on Penetration in Granodiorite*
3-inch 60° Cast-iron Cones; 100 Per Cent Blasting Gelatin

Shot No.	Taper		Spall, In.	Depth of Hole, In.	Total Depth, In.	Diameter of Hole, In.	Remarks
	Bottom	to Top					
1	4/32	3/32	9	4	13	1	
2			1½	8½	10	1¼	
3			3	7	10	1½+	
					Av. 11		
1	5/32	3/32	3	8½	11½	1¼	
2			4	11	15	1¼	
3			8	2	10	1¼	
					Av. 12.2		
1	6/32	3/32	4	8	12	1½	
2			2	10	12	1½	
3			8	0	8		
					Av. 10.7		
1	7/32	3/32	4	7	11	1½	
2			2	8	10		
3			2	7	9		Large spall area
					Av. 10		
1	5/32	3/32	2	5	7		Large spall area
2			4	8	12		
3			1	10	11		Bottom of old hole
					Av. 10		

of this increased effect are probably threefold: (1) The action of rebounding gas molecules from the inside surface of the case has an effect on more complete and perfect detonation of the explosive, and ensuring propagation of the high velocity of detonation of 100 per cent blasting gelatin; (2) much more of the force of the explosion itself is directed toward the point of least resistance, the cavity end of the charge. This has a result of (3) more of the explosive force being directed against the cavity liner itself to give a stronger jet.

These hypotheses are borne out by present theories on the behavior of molecules in explosion gases. Lawrence[3] says, "During this time (of detonation, one microsecond) at the pressures and temperatures involved the number of collisions is of the order of ten million per molecule—." From this picture we can easily see that if a portion of these high-speed molecules, which ordinarily escape in all directions, rebound from the walls of a rigid or semirigid case, they are unquestionably instrumental in effecting a more complete explosion as well as in affecting the factors mentioned above.

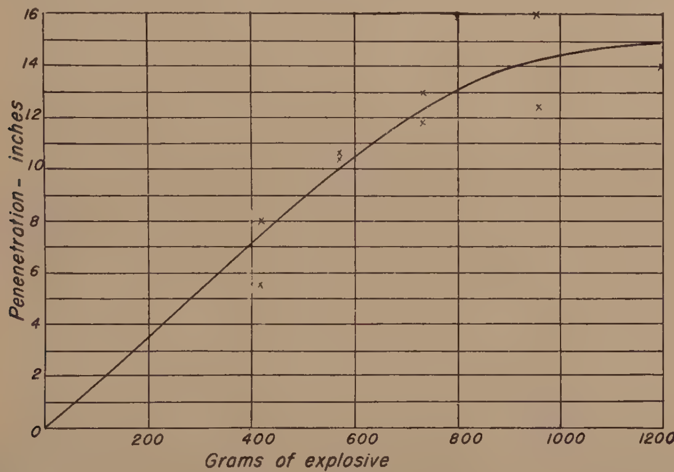

FIG 4—EFFECT OF AMOUNT OF EXPLOSIVE IN 3-INCH DIAMETER CONICAL CAVITY CLOSED CHARGE ON DEPTH OF PENETRATION IN SOLID GRANODIORITE.[2]
Standoff 2 in.: explosive 60 pct N.G. dynamite.

TABLE 3—*Effect of Confinement of Charge in Cast-steel Cases on Penetration in Solid Granodiorite*

3-in 60° Cast-iron Cones; 100 Per Cent Blasting Gelatin

Shot No.	Thickness of Case, In.	Spall, In.	Hole Depth, In.	Total Depth, In.	Diameter of Hole, In.
1	¼	3½	4	7½	1
2	¼	2	3	5	1
3	½	3	4	7	1½
4	½	2	7	9	1½
5	1	2	8	10	2
6	2	1	8	9	3
7	2	2	8	10	3
8	3	3	7	10	3

Metals Used in Cavity Liners

Table 4 gives the results of tests conducted with different metals used as 2-in. hemispherical charge liners.

All of the hemispherical liners (with exceptions noted) used in the present tests were made as unmachined castings. Bell-shaped aluminum castings were used to hold the explosive charge. Metals of higher melting points were annealed.

There is no apparent relationship between the melting points of the metals

and their performance in producing Munroe jets. Brittleness, hardness, cohesiveness and other properties affect their performance to a greater extent than melting point. Lead, which is the softest and has the lowest melting point of the metals tested, produced lower penetration values than the other metals. Aluminum, which has a higher melting point than zinc, gave lower penetration values than the latter. This would lead to a possible conclusion that brittleness is one of the desirable characteristics of a metal for shaped-charge cavity liners. An aluminum alloy, which possessed a much higher tensile strength than any of the other metals, gave the best performance of any of them.

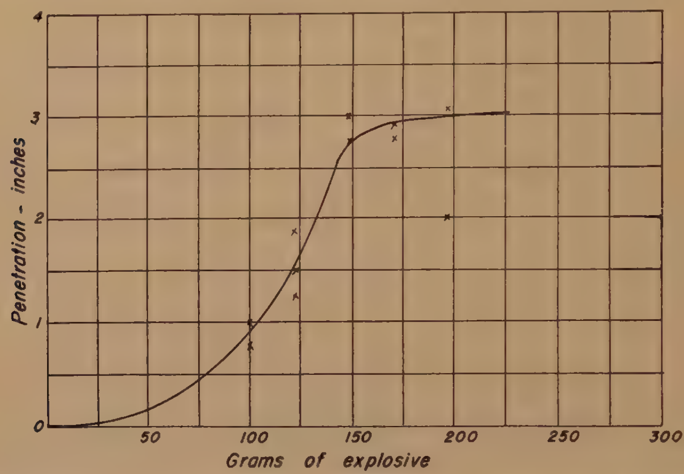

FIG 5—EFFECT OF AMOUNT OF EXPLOSIVE IN CHARGES WITH 1¾-IN. CAST-IRON HEMISPHERICAL LINERS ON DEPTH OF PENETRATION IN STEEL PLATES.[2]
Standoff 2½ in.; explosive 60 pct N.G. dynamite.

Two cast-iron hemispheres, rough and unannealed, were fired, but they produced such irregular jets that their values were included only as a matter of relative interest. As has been pointed out,[2] cavity liners must be free from stresses as well as geometrically symmetrical. These two liners were not annealed, hence they did not give optimum performance, probably because of residual internal stresses.

Thus it appears that the metals that possess a fortuitous combination of hardness, brittleness, cohesiveness and ability to flow uniformly when subject to high pressures are among the metals that will give the best results in producing Munroe jets.

Inert Disks

In an effort to change the shape of the detonation wave from plane to curvilinear, inert aluminum disks 1½ in. in diameter and ⅜-in. thick were inserted in 3-in. charges at varying distances from the top of the charge. The only effect produced was to impede the mechanism of jet formation when the disk was placed close to the cavity liner.

Concrete Cases

In secondary breakage in open-pit mining, it would be desirable to use material for cases other than metal in order to avoid danger to personnel from shrapnel from metal cases. A few concrete cases of the same dimensions as the aluminum cases for hemispherical charges were made of portland cement and fine sand with a fiber binder. Table 5 gives the results of three firings of this type of

TABLE 4—*Penetration of Two-inch Hemispherical Charges in Steel Plates with Liners of Different Metals and Alloys*

3-inch Standoff; Cast Aluminum Cases Loaded with 100 Per Cent Blasting Gelatin

Metal	Shot No.	Number of Plates	Depth of Hole, In.	Diameter of Hole, In.	Remarks
Al	1	8	2	2	
	2	12+	3	2	
	3	12+	3	2	
		Av. 10.7			
Pb	1	7	1¾	1½	Irregular jet
	2	12	3	1¾	
	3	9½	2⅜		
		Av. 9.5			
Zn	1	12	3	1¾	
	2	12	3	1¾	
	3	11½	2⅜		
		Av. 11.8			
Al alloy	1	10	2½	2	
	2	dud			
	3	16+	4+	1½	
	4	15½	3⅞	1¾	
	5	9+	2⅜	1¾	
		Av. 12.8			
Fe Sand-cast unannealed	1	8	2	1¼	Very irregular jet
	2	12	3	1¼	Very irregular jet
		Av. 10			

charge fired for effect on steel plates. The average penetration (8.7 in.) was considerably below that (12.8 in.) for aluminum cases. Later tests on iron ore also proved these cases of concrete to be relatively unsatisfactory.

TABLE 5—*Effect of Confinement of Concrete Cases on Penetration of Two-inch Hemispherical Charges in Steel Plates with Aluminum-alloy Liners and 100 Per Cent Blasting Gelatin*

No.	Number of Plates	Depth of Hole, In.	Diameter of Hole, In.
1	7	1¾	2
2	12	3	2
3	6½	1⅝	2
	Av. 8.7		

Confinement tests, previously discussed, showed conclusively that steel, which possesses high tensile strength, makes a satisfactory case material. The total effect of confinement, however, probably is related to mass as well as to tensile strength.

Strength of Explosive

It is evident that an explosive with greater strength than another will be more effective in producing Munroe jets. Tests were conducted with three kinds of explosives (Table 6) to determine the approximate relationship between velocity of detonation (here assumed to be roughly proportional to the explosive strength) and penetration effect. While only a few shots were fired, the results obtained from them give a very good indication that the relationship is approximately linear, although for a given increase in

TABLE 6—*Penetration of Three Kinds of Explosives in Steel Plates Using Two-inch Hemispherical Charges with Cast Aluminum Cases, Aluminum-alloy liners and Three-inch Standoff*

Shot No.	Number of Plates	Depth of Hole, In.	Diameter of Hole, In.
45 Pct Gelamite: Velocity of Det., 8,500 Ft per Sec			
1	9	2½	1¼
2	5	1¼	1¼
3	5	1¼	1¼
	Av. 6⅓		
60 Pct N.G. Dynamite: Velocity of Det., 19,000 Ft per Sec			
1	5+	1⅜	2
2	11+	2⅞	2
3	8+	2⅛	1⅞
	Av. 8		
100 Pct Oilwell Explosive: Velocity of Det., 26,200 Ft per Sec			
1	10	2½	2
2	dud		
3	16+	4+	1½
4	15½	3⅞	1¾
5	9+	2⅜	1¾
	Av. 12.8		

velocity of detonation, say, if the velocity of detonation is doubled, the penetration is not doubled, but is increased by some factor that is less than one times the velocity of detonation. More complete tests may show that a curve plotted between velocity of detonation and penetration is an exponential one, but it is believed at present that the curvature for the range of velocities tested is rather slight. This also probably would hold true only where explosive strength is closely related to velocity of detonation.

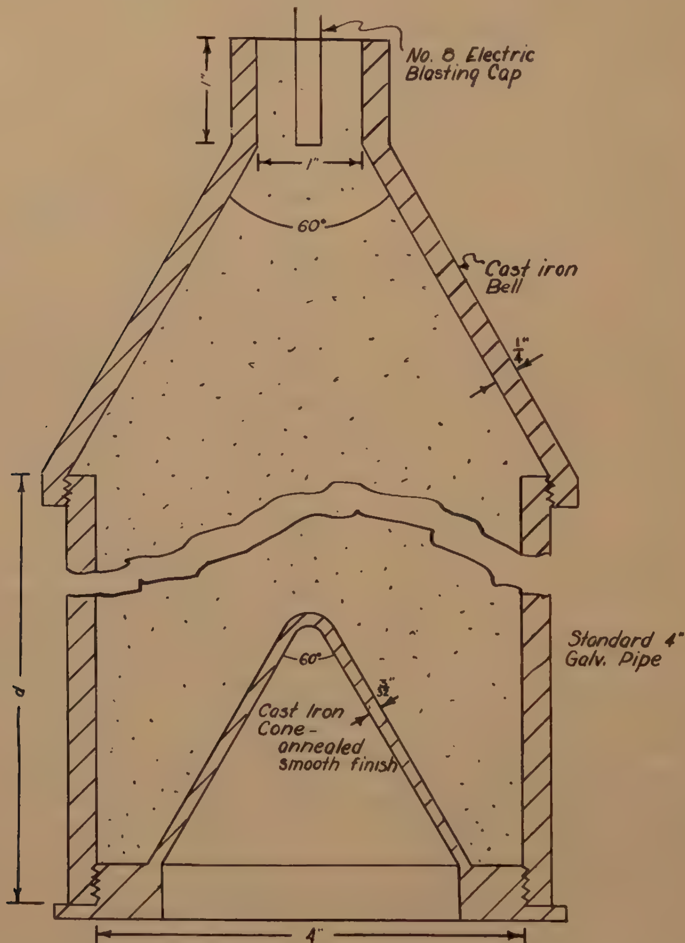

FIG 6—SHAPED CHARGE WITH CAST-IRON CONE LINER, CAST-IRON BELL AND GALVANIZED PIPE CASE.

Breakage Tests on Concrete Blocks

Breakage tests were carried out on three sets of graded sizes of concrete blocks (Table 7). Based upon previous tests on granodiorite, the blocks were made larger in horizontal dimensions than vertical. Where a shaped charge is placed so that the axis of the charge is parallel to one of the longest dimensions of a rock, the resultant breakage is limited to spalling the block in all directions, leaving it pyramidal in shape with the apex of the pyramid under the point of

application of the charge. This is well illustrated in Fig 17, an 18-in. cube of concrete broken by a charge loaded with 40 pct Gelamite. If the block had been show the relationship between breaking power of a 2-in. charge and the increase in size of blocks. The curves for increase of volume by increase with vertical dimen-

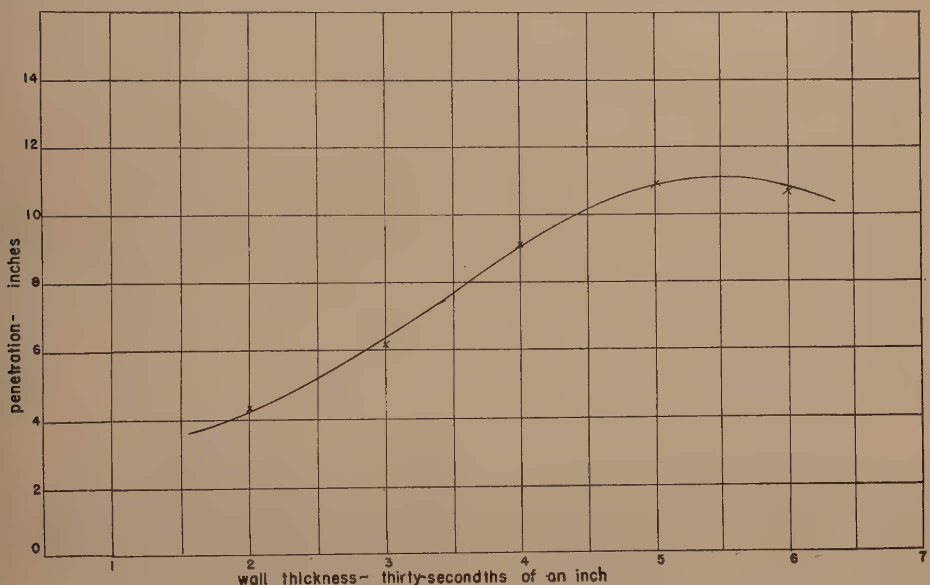

Fig 7—Effect of charge in cone-wall thickness (uniform wall) on penetration in granodiorite.

Explosive: 100 pct blasting gelatin. Cones: cast-iron, 3-in. i.d. with 60° apex angle.

larger on its vertical axis, probably it would have failed to break along the two vertical planes as the 18-in. cube did.

The charges used were 2-in. hemispherical with aluminum-alloy liners, aluminum cases, loaded with 100 per cent blasting gelatin, 60 pct N.G. dynamite and 45 pct Gelamite. Fifty-four charges were used, each third of them being loaded with one kind of explosive. To evaluate the relative breakage effect the following breakage-index scale was used:

Index No.	Type of Breakage
0	No breakage, penetration only
1	Block cracked only
2	Block completely broken and separated into four quadrants
3 to 8	Gradations between 2 and 9
9	All pieces 10 lb or smaller
10	All pieces 5 lb or smaller

The results of tests on the 54 blocks are shown in Table 7. Selected groups of these results are plotted in Fig 9 to

sion only are much steeper than those for increase of volume with increase of horizontal dimensions; that is to say, a given size of charge will completely break much heavier pieces of rock if they are relatively flat than if they are nearly cubical, or if the charge is placed parallel to one of the longer dimensions of the rock.

Results of breakage tests might also be evaluated on the basis of Rittinger's law of crushing, that the energy absorbed in crushing rock is proportional to the area produced in the process. Many tests in the field of ore dressing have proved it to be the fundamental principle of rock breakage. The amount of energy required for producing finely ground materials increases very rapidly with diminishing particle size; for example, the relative surface of minus 200-mesh (0.074-mm)

TABLE 7—*Size and Weight of Concrete Blocks Used in Breakage Tests with Breakage Indices Tabulated for Three Strengths of Explosive*

Horizontal Dimensions, In.	Volume, Cu Ft	Weight, Lb	Breakage Number		
			45 Pct Gel.	60 Pct N.G.	100 Pct B.G.
12 In. Thick					
12 × 12	1.00	150	9	10	10+
18 × 18	2.25	338	8	9	10
24 × 24	4.00	600	6	8	9
30 × 30	6.25	938	4	7	8
36 × 36	9.00	1350	2	6	7
14 In. Thick					
14 × 14	1.59	238	8	9	10
18 × 18	2.56	384	7	8	9
24 × 24	4.65	696	5	7	8
30 × 30	7.25	1087	3	6	7
36 × 36	10.47	1572	1+	5	6
16 In. Thick					
16 × 16	2.37	356	7	8	9
24 × 24	5.33	798	5	6	8
30 × 30	8.30	1243	3	5	7
36 × 36	12.00	1800	1	4	6
18 In. Thick					
18 × 18	3.38	507	6	7	8
24 × 24	6.00	900	4	5+	7
30 × 30	9.40	1410	2	4	6
36 × 36	13.50	2060	0	3	5

material is 512 times that of minus 1-in. material (26.67-mm).

Breakage tests were conducted in the open, consequently it was not possible to recover all of the solid products of the explosions. "New area" formed was determined for broken pieces of 15 lb or over. A comparison of the breakage of 18 by 36 by 36-in. blocks by three explosives can be used as a guide in fixing the limits of breakage. Forty per cent Gelamite in a 2-in. charge failed to break a block of the size indicated above. Sixty per cent N.G. dynamite broke an 18 by 36 by 36-in. block into four pieces and 100 pct blasting gelatin shattered and separated the same size block into several large fragments. (See Figs 10 to 17.)

These figures show clearly the difference in effect of higher-strength explosives and the advantage gained in using them in secondary breakage. Figs 10 and 12 illustrate the added breaking power of 60 pct N.G. dynamite over 45 pct Gelamite, while Figs 12 and 13 show the difference in effect of 100 pct blasting gelatin and 45 pct Gelamite. Figs 14 and 15 give a comparison of 60 pct N.G. dynamite and 100 pct blasting gelatin. The possible breaking power of the jet is assisted considerably by the impact blow of the shock wave of the explosion.

It is believed that the total available energy from a Munroe jet that may be utilized in breakage is fairly constant for a given design of charge. Assuming that the Munroe jet energy and the total impact energy of the explosive is constant for a charge of given size, as the size of the target blocks is increased more and more of this energy is absorbed in penetration than in breakage, until limiting dimensions of depth and/or width of block are reached where the total energy of the jet is used up in spalling and penetration only. Under conditions where only a hole is produced by a shaped charge, more energy is also lost in heat and in producing very fine material. In a small rock, as the jet commences to penetrate the total cohesion of the rock around the hole is not sufficient to prevent breakage and the rock is forced apart. For large blocks, energy that is used for breakage in small blocks is dissipated largely as heat of friction. For progressively smaller blocks, the proportion of energy absorbed in breakage is believed to increase slightly because of the small relative total cohesiveness of smaller blocks.

In regard to the proportion of energy used in penetration and that used in breakage, if the volume of the hole produced in a block of concrete were considered to have contained one pound of concrete before displacement, the energy

used in crushing it would be many times that used in breaking the block. There is good evidence to substantiate this belief. The size of the larger portion of pany's iron mine in southern Utah. The thickness of the blocks of ore was from 18 to 24 in. Their lateral dimensions were from 2½ to 4½ ft. These

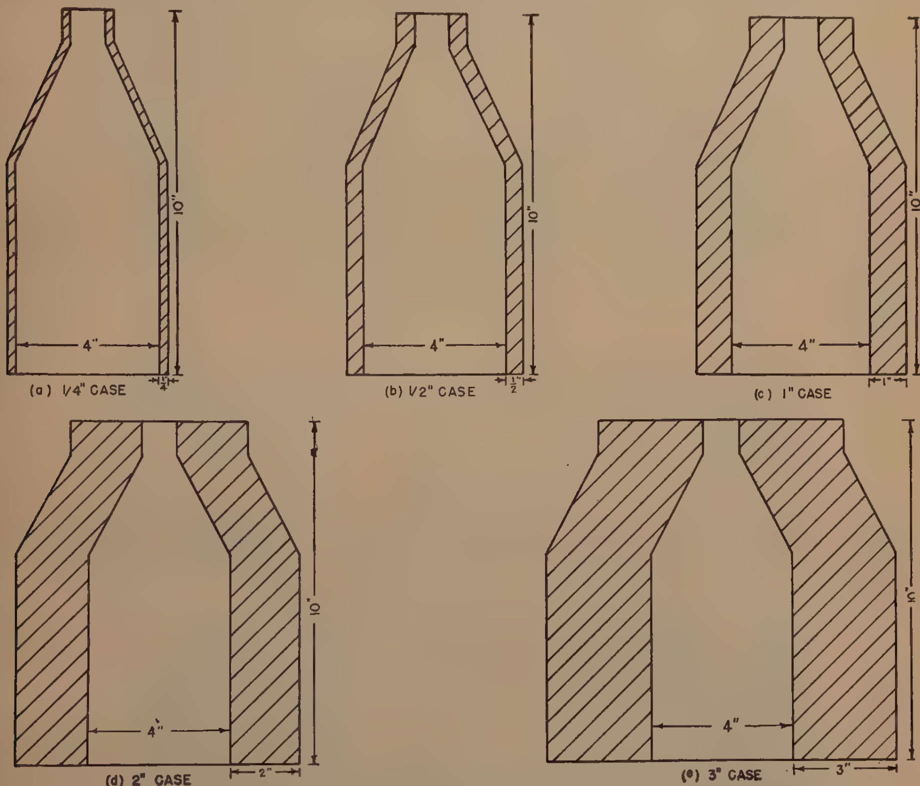

FIG 8—CAST-STEEL CASES USED IN CONFINEMENT TESTS WITH CAST-IRON 60° CONES.

a. ¼-in. case.
b. ½-in. case.
c. 1-in. case.
d. 2-in. case.
e. 3-in. case.

dust created by Munroe penetration into solid rock is probably finely pulverized material in the neighborhood of 200-mesh. The specific surface of one pound of 200-mesh material is very close to 250 sq ft. The largest "new area" created in secondary breakage in the present tests was in the neighborhood of 25 sq ft, which is less than one tenth that of the dust.

Shaped Charges and Iron Ore

A few charges were tried on two boulders of iron ore at the Columbia Steel Company's boulders were completely broken by 2-in. charges. The results indicate that larger charges are necessary to break the boulder iron ore than would be needed for material like rock and concrete. It is suggested that well-placed shaped charges may also be useful in freeing clogged chutes in block-caving systems of mining.

Summary and Conclusions

1. Optimum wall thickness for conical cavity liners is dependent upon apex

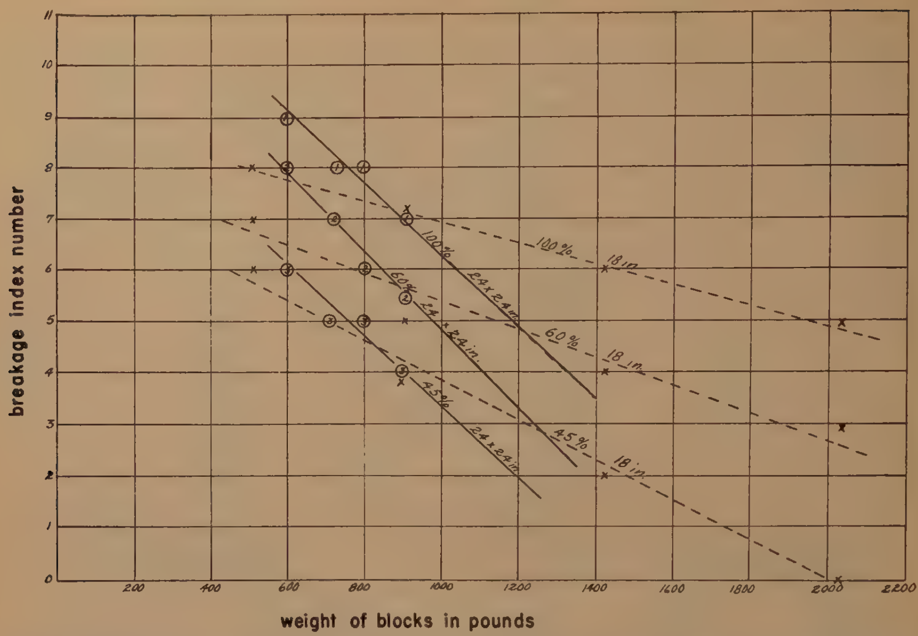

FIG 9—RELATIVE BREAKING POWER OF THREE TYPES OF EXPLOSIVE WITH CHANGE IN SIZE OF CONCRETE BLOCKS IN: (1) HORIZONTAL DIMENSIONS (DOTTED LINES) AND (2) VERTICAL DIMENSIONS (UNBROKEN LINES).

Explosives used: 100 pct blasting gelatin, 60 pct N. G. dynamite and 45 pct Gelamite.

FIG 10—BLOCK MEASURING 12 BY 36 BY 36 INCHES BROKEN BY 2-INCH HEMISPHERICAL CHARGE CONTAINING TWO STICKS OF 45 PER CENT GELAMITE.

Block was partially cracked into quadrants.

angle as well as base diameter of the charge.

2. Tapered-wall cones proved more effective than cones with walls of uniform thickness for the thicknesses tested herein.

3. The effect of confinement of the explosive charge was to increase both the diameter and the depth of hole produced.

4. The physical and mechanical characteristics of the metal in the cavity liner have a marked effect upon the performance of a shaped charge.

FIG 11—BLOCK 12 BY 36 BY 36 INCHES BROKEN BY 2-INCH HEMISPHERICAL CHARGE CONTAINING TWO STICKS OF 60 PER CENT N. G. DYNAMITE.

Block is of same size as that shown in Fig 10, but is completely broken by the higher strength explosive.

FIG 12—BLOCK 16 BY 36 BY 36 INCHES AFTER SHOOTING WITH 2-INCH HEMISPHERICAL CHARGE WITH 45 PER CENT GELAMITE.

Result: slight penetration by jet and block barely cracked into quadrants.

FIG 13—CONCRETE BLOCK 16 BY 36 BY 36 INCHES BROKEN BY 2-INCH HEMISPHERICAL CHARGE LOADED WITH 100 PER CENT BLASTING GELATIN.

Block well penetrated by jet and pieces well scattered.

FIG 14—CONCRETE BLOCK 18 BY 36 BY 36 INCHES BROKEN BY 2-INCH HEMISPHERICAL CHARGE LOADED WITH 60 PER CENT N. G. DYNAMITE.

Breakage similar to that for same size block with 3-inch charge with 45 per cent Gelamite (Fig 16).

FIG 15—CONCRETE BLOCK 18 BY 36 BY 36 INCHES BROKEN BY 2-INCH HEMISPHERICAL CHARGE LOADED WITH 100 PER CENT BLASTING GELATIN.

Same size blocks as Figs 14 and 16, but with greater degree of breakage.

FIG 16—CONCRETE BLOCK 18 BY 36 BY 36 INCHES BROKEN BY 3-INCH HEMISPHERICAL CHARGE CONTAINING 4½ STICKS OF 45 PER CENT GELAMITE.

Shows same degree of breakage as for smaller charge containing 60 per cent N. G. dynamite (Fig 14).

5. Cavity effect of shaped charges is roughly proportional to the velocity of detonation of the explosive used where detonation velocity is closely related to the strength of the explosive.

6. Total breakage energy available in a given shaped charge appears to be fairly constant, decreasing somewhat for larger blocks of concrete.

7. A few small shaped charges proved effective in reducing 2½-ton blocks of iron ore that were too large to be handled by power shovels.

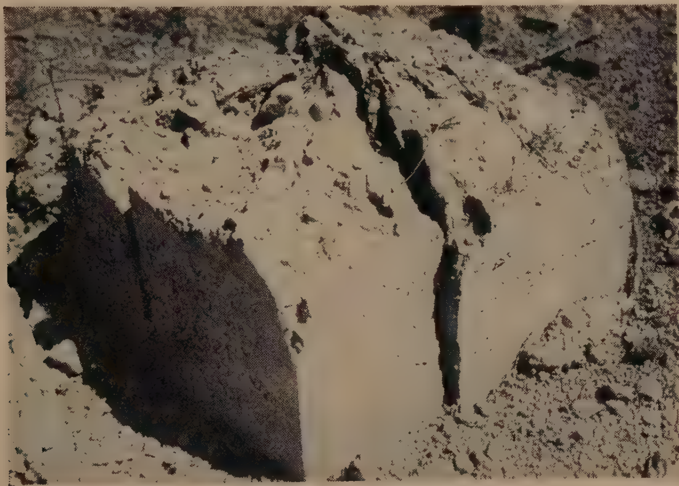

FIG 17—AN 18-INCH CONCRETE CUBE BROKEN BY 2-INCH HEMISPHERICAL CHARGE LOADED WITH 45 PER CENT GELAMITE.

Note tendency to break to pyramidal shape when depth of block approaches horizontal dimensions in magnitude.

ACKNOWLEDGMENT

Grateful acknowledgment is made to Dr. A. Ray Olpin, President of the University of Utah, and to the Kennecott Copper Co., which made funds available for this research. A vote of thanks is also due the Whitmore Oxygen Co. and the Utah State Road Commission for use of lands under their control for field experiments.

This paper presents the results of experimentation carried out by George B. Clark, Mining Engineer, in cooperation with the Mining Department of the University of Utah in the summer of 1946, and is a continuation of a series of studies on the theoretical development and practical application of shaped explosive charges begun in October 1945.

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3. R. H. Lawrence: Mechanism of Detonation in Explosives. Reprint by Hercules Powder Co. from *Geophysics* (Jan. 1945) **9**.
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DISCUSSION

(E. D. Gardner presiding)

PHILIP B. BUCKY*—The people from Illinois have given us considerable data. I should like more of the reasoning behind the statement whereby it is concluded that the forces from

* Professor of Mining, Columbia University, New York, N. Y.

the charge will come together as they do and go forward from the shape of the cone.

The theory we have had up to now has been that the force of the explosive is exerted equally in all directions.

George B. Clark (author's reply)—I think that question could possibly be answered on the basis of hydrodynamic theory. The pressure that exists in the detonation wave as it proceeds through the explosive is very intense, much more so than that in the gases surrounding the explosive zone of action.

It is the pressures in the detonation wave itself, in the burning zone, which exert the pressure on the cast iron or other cavity liners, and these pressures are so intense that they cause the metal to flow with practically a negligible friction effect and it becomes a problem in hydrodynamics. If this problem is solved I think that one can mathematically justify the existence of the jet and evaluate its momentum and the various other factors such as impact and kinetic energy. Briefly, the effect of the cavity is to concentrate the force of a certain portion of the lower part of the column of explosive.

There is no increase in total energy of the explosive, but it causes a concentration of the force that results from the explosion.

V. H. Clarke*—Did you say that mud blasting, up to a ton and a half rock, was more effective than your shaped charge? Could you elaborate on that a little more? Was your testing with mud blasting very extensive?

George B. Clark—Our comparative tests between shaped charges and mud capping were rather few in number and about the only thing I can do is to restate what I stated a few moments ago: insofar as we know, mud capping is more effective in breaking concrete blocks of the size indicated in the foregoing discussion than are the conventional types of shaped charges.

E. D. Gardner†—With the same amount of explosives?

George B. Clark—With the same amount of explosive.

* Howe Sound Co., New York, N. Y.
† U. S. Bureau of Mines, Denver, Colo.

Philip B. Bucky—I believe the questioner wanted to know the reason, the logic, behind it. Do you have any logic to explain why mud blasting is better than shaped charges for that use?

George B. Clark—There are, as I mentioned in the beginning of the paper, a large number of variables which enter in the performance of a shaped charge.

In my previous statement concerning comparisons of mud capping and shaped charges, I purposely hedged because I feel, and I think others feel the same way, that we do not know all there is to be known about the possible effect of a cavity liner. I will say this, however, if we appraise the character of a jet based upon the present theory, we may get a clearer picture of its action in drilling holes and breaking rocks. First of all, the jet, if the present theory is correct, is a stream of very small microscopic particles of solid material. The effect, then, is similar to that of a stream from a hydraulic nozzle which is playing the stream into a bank of mud. It washes the material out of the bank but does not propagate too much of an expanding force in the mud bank itself.

The hydraulic jet does not have the property that an expanding jet of gases would have, which would probably account to some extent for the difference in effect that is obtained between a mud cap which deals the rock a very hard blow underneath the explosive where it is in contact with the rock and a jet which produces a very concentrated hydraulic action, that puts a hydraulic jet inside of the rock which has a limited property of expansion.

M. H. Hawkins*—I would like to ask you if you made any experiments without any liner at all, simply shaping the cavity.

As I understand it, you think the effect is due to metal particles, but if you use no liner at all, you would not have any metal particles; would you?

George B. Clark—That is correct. In that case, the jet which is formed is gaseous and will make a crater in the target of a very limited depth usually with a top diameter equal to the diameter of the cavity. Its penetration effect is very limited because of the fact that the gase-

* Newburgh, N. Y.

ous molecules may escape in all directions once they move out of the cavity.

A metal jet does not tend to expand after it leaves the cavity; it continues along a straight line.

H. L. WALKER*—Professor Clark has some photographs of the rock breaking qualities of shaped charges and adobe charges which have been fired in competition with each other, all other conditions remaining the same. These photographs definitely show that mud-capping does a better job, at least for the conditions of the experiment.

One possible type of design Professor Clark did not comment on is a charge having the form of a spherical sector with a large radius. If the flat portion of the sector were placed against the work there would probably be an increase in energy utilized in breaking rock over that absorbed from a shaped charge. The shaped charge tends to penetrate without fracturing.

*University of Illinois, Urbana, Ill.

Behavior of Metal Cavity Liners in Shaped Explosive Charges

By George B. Clark* and Walter H. Bruckner,† Members AIME

(New York Meeting, March 1947)

Since the end of World War II interest has been increasing in the use of shaped charges in the mining industry and in other industries using explosives for blasting purposes. Shaped charges employ the principle known as the "Munroe effect," which was discovered by Charles E. Munroe more than 50 years ago (in 1888). Details of their design have been explained elsewhere.[1,2,3] Fig 1 shows the essential features of design of two types of shaped charges (with conical and hemispherical cavities) and a schematic sketch of their action upon detonation.

The following discussion deals with the behavior of the metal in the cavity liners when they are subjected to intense pressures exerted when the explosive charge is detonated. Among the conclusions reached in research on shaped explosive charges, the following have been established concerning cavity liners:[1,2]

1. The optimum wall thickness of a conical cavity liner is dependent upon the apex angle of the cone as well as on the base diameter of the charge. Acute apex angles require thinner walls for optimum performance and more obtuse angles require thicker walls for the same base diameter.

2. Cones were more effective in forming a penetrating jet when the walls were tapered; that is, the thickness of the wall of the cone increased from the apex down.

3. The physical and mechanical properties of metals have a marked effect upon their performance as cavity-liner material. Boiling point, ductility, malleability, tensile strength, and hardness are among the properties that influence the effectiveness of a metal used as a cavity liner. Lead, for example, makes a wide, flat crater in steel plates. Aluminum makes a deeper crater than lead, and an aluminum alloy having a high tensile strength makes a deeper hole, but slightly smaller in diameter. Cast iron makes a deep, narrow hole.

These findings, together with the following analysis under Metallographic Survey of Slug, offer solutions to many of the problems involved in solving the mechanism of the formation of Munroe jets.

It has been fairly well established that cavity liners collapse in a manner similar to that indicated in Fig 2. Conical liners are known to collapse upon themselves while hemispherical liners are believed to turn inside out in the process of jet formation. The first has been definitely proved by recovery of collapsed portions of cones, while liners from partly detonated charges show that hemispheres may turn inside out. Fig 3 shows a sketch of a recovered slug from a 6-in. shaped charge using a 45° cone made of cast iron. (This size of charge drilled holes up to 3 ft in depth in solid granodiorite.)

Manuscript received at the office of the Institute Dec. 16, 1946. Issued as TP 2158 in Metals Technology, Aug. 1947 and Mining Technology, May 1947.

* Assistant Professor of Mining Engineering, University of Illinois, Urbana, Illinois.

† Research Assistant Professor of Metallurgical Engineering, University of Illinois.

[1] References are at the end of the paper.

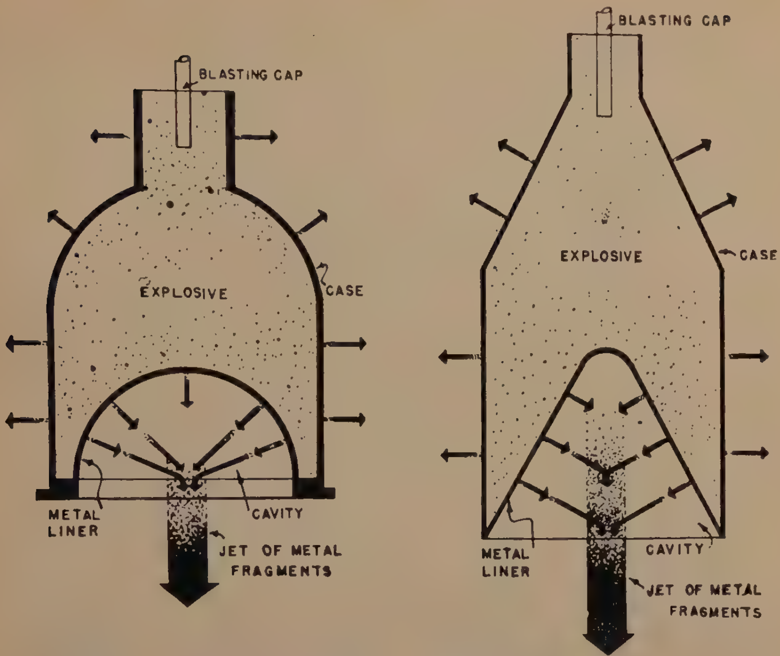

FIG 1—SKETCHES OF SHAPED CHARGES WITH (*a*) HEMISPHERICAL AND (*b*) CONICAL CAVITIES SHOWING MECHANISM OF FORMATION OF MUNROE JET.[1]

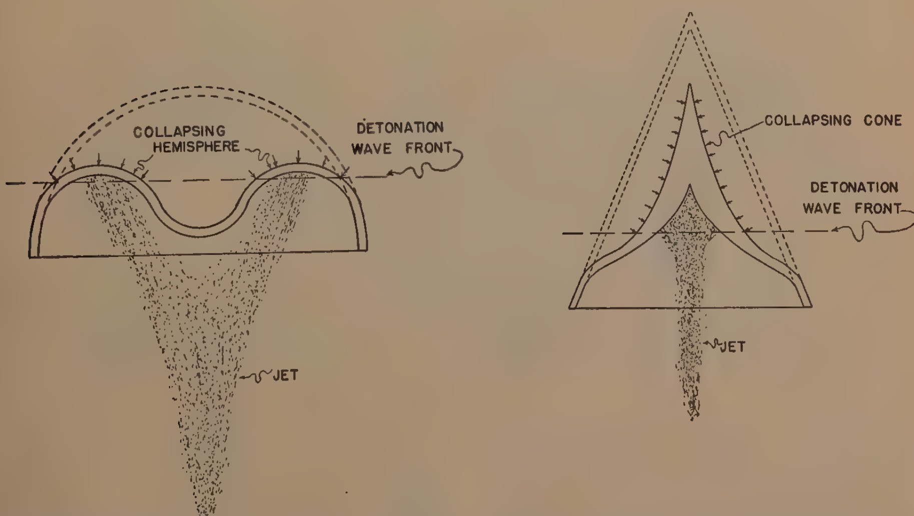

FIG 2—DIAGRAMMATIC SKETCH OF COLLAPSE OF CAVITY LINERS SHOWING APPROXIMATE POSITION OF DETONATION WAVE AND FORMATION OF JETS FROM REGIONS OF GREATEST COMPRESSION IN THE METAL.

Metallographic Survey of Cast-iron Slug

The cast-iron slug was sectioned on a diametral plane through the original axis

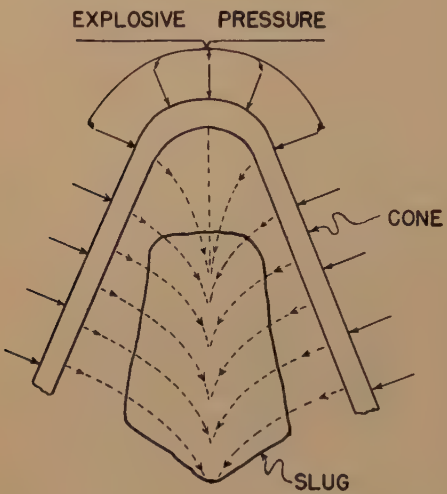

Fig 3—Collapsed slug from six-inch cast-iron cone and portion of original cone from which it came.

Pressure is represented by arrows normal to the surface and curves inside the cone are probable paths of particles from the inside surface of the cone.

of the cone. This plane of metal received a metallographic polish and was etched with 5 pct nital to give the macrograph in Fig 4 at a magnification of approximately 2½ diam.

A piece of the original iron casting from which the cast-iron cone had been made was also available. It was polished and etched to observe the original microstructure of the slug. A typical representation of this microstructure is shown in Fig 6. The structure shows the presence of straight-sided masses of graphite from which graphite flakes radiate (type C, ASTM). There appears to be little or no free ferrite, thus the microstructure consisted mainly of pearlite and graphite. However, it was reported that the cone had been heated after machining to a cherry red (temperature approx. 1400°F) and furnace-cooled. There was no specimen available to determine the effect on the microstructure of the heat-treatment given the iron. The structure of the recovered slug near the top outer edges and other portions representing regions of least plastic deformation are comparable with the microstructure shown in Fig 6, thus the heat-treatment was ineffective in producing any major change in microstructure.

A survey of the microstructure on the entire plane of polish of the slug illustrated in Fig 4 was made. A record was made of characteristic regions at a magnification of 100 and 500 diam. In Fig 5 a copy of Fig 4 is given with circled areas, which are lettered to correspond with the micrographs that follow in Figs 7 to 14. The micrographs are oriented the same as Fig 5 with respect to top and bottom and were taken in the circled regions shown in Fig 5.

Fig 7 shows a region at the center, top of the slug, which had suffered considerable plastic deformation, as indicated by the "lining up" of the graphite flakes. Where the largest amount of flow or deformation took place, shown at the right side at the bottom of Fig 7, the graphite was surrounded by free ferrite and the pearlite areas were compacted. Fig 8 shows the manner in which the flow took place in this particular region. In the area in the lower right-hand side of Fig 8, the graphite, pearlite and ferrite are compacted into flow layers adjacent to a region of practically undisturbed lamellar pearlite at the top left. The structure at the right consists of free ferrite, graphite and compacted pearlite. Fig 9 shows a region in which the center field has a large amount of ferrite from which the graphite was almost completely removed. The ferrite shows a new granular structure resulting from recrystallization and the pearlite is distributed throughout as a fine dispersion. The pearlite in the surrounding areas appeared to have been partially spheroidized. The

FIG 4—CAST-IRON SLUG. × 2½.

recrystallization of the ferrite and the partial spheroidization of the pearlite indicate that a temperature in the neighborhood of 1300°F was attained in the particular region.

Figs 10, 11, and 12 show a series of micrographs that form a sequence along a horizontal line shown in Fig 5. The sequence is shown by the deformation of the ferrite in Fig 10, the partial recrystallization of the deformed ferrite in Fig 11 and the complete recrystallization of the ferrite and the partially spheroidized pearlite in Fig 12. The sequence suggests

that a temperature gradient may have been established in the slug with the temperature maximum along the center line of the slug and with the maxima

FIG 5—SLUG WITH AREAS LETTERED TO CORRESPOND WITH MICROGRAPHS OF FIGURES 7 TO 14. × 2½.

increasing from top to bottom of the slug shown in Figs 4 and 5. It was also observed that in going toward the center of the slug the microstructure progressively contained less graphite, while in the central area of the slug the graphite appeared to have been squeezed out into large areas of agglomerated, practically pure graphite. In the sequence of Figs 10, 11, and 12 the direction of flow progressively changed until, as in Fig 12, it was almost parallel with the axis of the original cone.

Figs 13 and 14 show the microstructures near the bottom of the slug. Fig 13 shows a region that consists of recrystallized ferrite and recrystallized pearlite plus some graphite, while Fig 14 shows only recrystallized ferrite and spheroidized pearlite plus graphite. The areas of recrystallized pearlite in Fig 13 must have attained a temperature considerably in excess of 1300°F, below which temperature the deformed ferrite alone will recrystallize.

Near the bottom of the slug there were a number of fractures along the direction of flow. The fractures invariably went through areas of graphite that represented planes of weakness.

The extreme ductility exhibited by the interior regions of the slug is unusual for a gray cast-iron composition representative of the cone material. In tension the gray iron is notoriously a material of low strength and brittleness. Under the compressional stress of the shaped charge and the rapid application of the stress, the ductile behavior of the cast iron may have been enhanced by the heat developed internally by friction.

Collapse of Cavity Liner

If it is assumed that approximately 25 pct of the cavity liner was ejected to form the jet, Fig 3 shows the approximate length of the original cone that is represented by the slug. The macroscopic flow structure indicates that the paths of the grains remaining within the slug are hyperbolic in shape; that is to say, the grains appear to have been acted upon by two forces, one perpendicular to the wall of the original cone and one parallel to the axis of the cone, the velocity due to the latter increasing in magnitude as the cone collapsed and the velocity due to the force normal to the surface being changed into a velocity parallel to the axis.

The kinetic pressure against the outside of a liner is caused by the intense bombardment of molecules of the explosive gases. These molecules probably do not penetrate more than one or two atomic layers into the surface of the metal liner.

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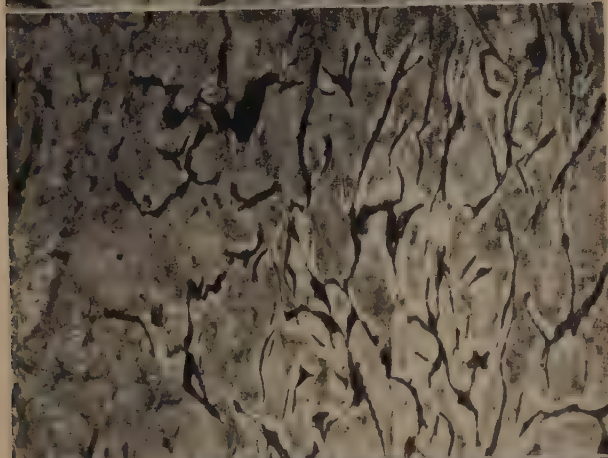

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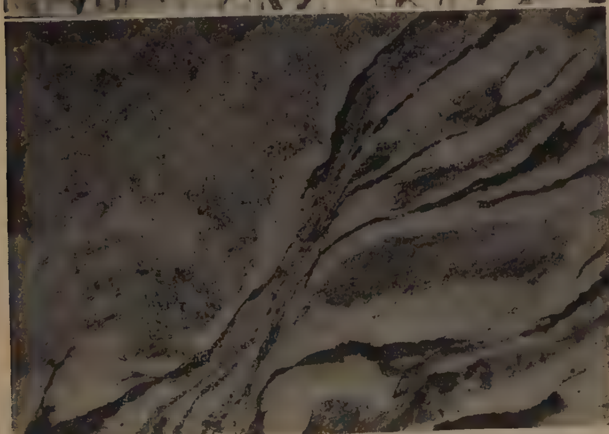

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FIG 6—TYPICAL MICROSTRUCTURE OF METAL FROM WHICH SLUG WAS MADE. × 500.
FIG 7—REGION AT CENTER TOP OF SLUG. × 100.
Shows lining up of graphite flakes. Right bottom, graphite surrounded by free ferrite and pearlite areas.
FIG 8—MANNER OF FLOW IN REGION AT LOWER RIGHT OF FIGURE 7. × 500.
Free ferrite, graphite and compacted pearlite at right.

Original magnifications given. Reduced one fourth in reproduction.

Their intense impact pressure, however, causes the wall of the liner to be forced in toward the axis of the cone. A study of the microstructure of the collapsed properties of the metal, together with wall thickness, type of explosive, and other factors.

Microscopic examination of the col-

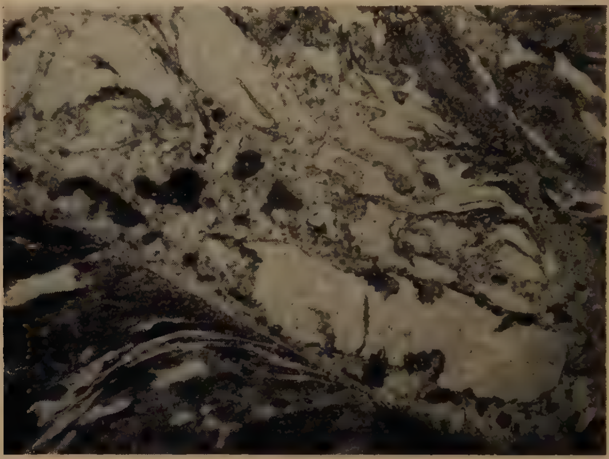


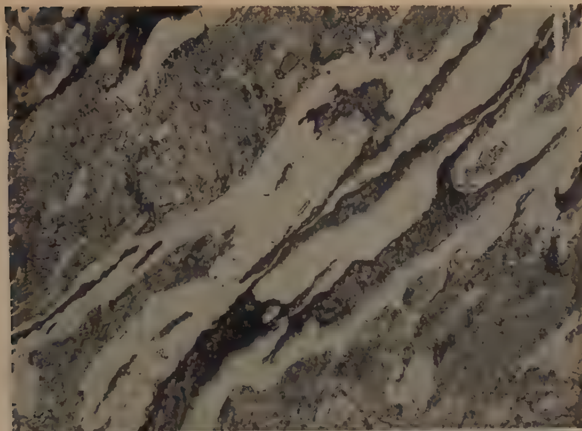
FIG 9—FERRITE FROM WHICH GRAPHITE WAS ALMOST COMPLETELY REMOVED. × 500. Original magnification given. Reduced one fourth in reproduction.

slug of cast iron (Figs 4 to 14) shows that the more malleable constituents of the iron have been literally squeezed into a new structure in the direction of the axis of the slug. The total picture of the mechanism of jet formation then must include the effect of the impact of the explosive gases plus the compressive forces set up in the metal. It was concluded that in a cone the grains are in effect acted upon by two sets of compressional forces. The structure of the cast iron in Figs 4 to 13 would lead to the same conclusion; that is, that compressive forces in the collapsing liner cause the ejection of particles from the inside surface, and these particles travel in hyperbolic paths asymptotic to or coincident with the axis of the cone.

The laws that govern the division of liner material into slug and jet are not quite clear. The contact angle of impact of the collapsing walls would have a marked influence on the separation as well as the physical and mechanical

lapsed slug of cast iron shows that the metal near the center of the slug reached much higher temperatures than the metal near the outside. At the center of the slug the metal is very fine grained, grains of one and two microns in diameter being very common. As a cone collapses the metal on the inside "layers" of the cone wall, which is the metal that goes to form the jet and the center of the slug, are subject to greater heat of deformation than the outside layers of the cone. The internal heat of friction of these layers is greater than that for the outside layers, with a resulting higher temperature and greater fluidity (plasticity) in the center.

The mechanism involved probably could be described as "extrusion through a gradient orifice"; that is, the collapsing cone furnishes the extruding agent, the extruded material and the "orifice" through which the material is forced. We would have, in effect, a syringe with a self-contained moving conical metal "bulb"



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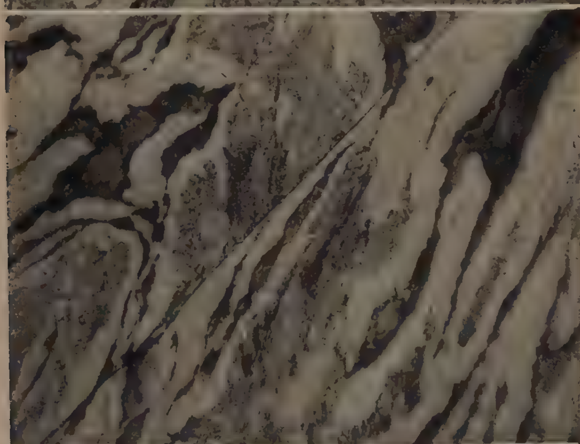

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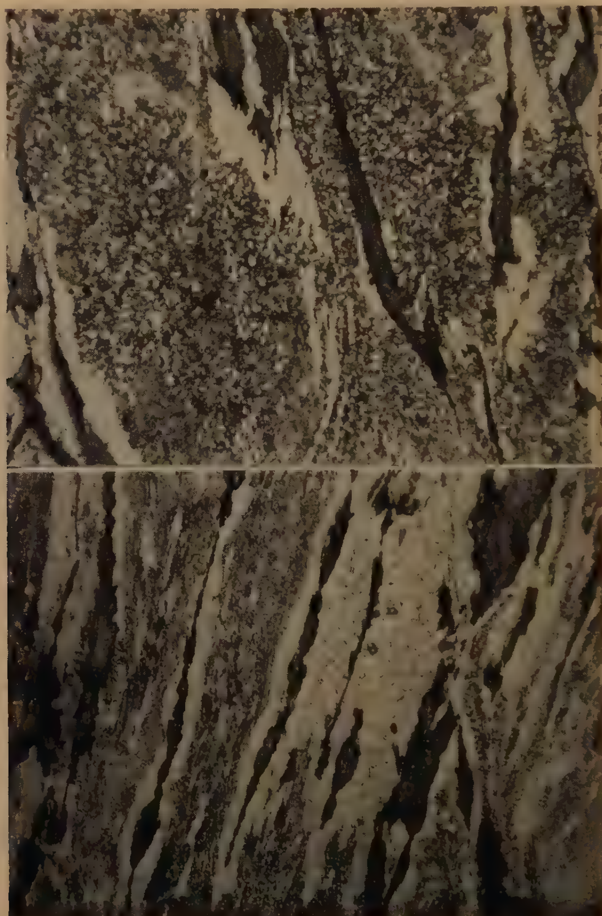
FIGS 10–12—SERIES OF MICROGRAPHS ALONG HORIZONTAL LINE IN FIGURE 5. × 500
Fig 10, deformation of ferrite.
Fig 11, partial recrystallization of deformed ferrite.
Fig 12, complete recrystallization of ferrite and partially spheroidized pearlite
Original magnification given. Reduced one fourth in reproduction.

subject to compressional forces normal to its walls. The "bulb" consists of metal of increasing fluidity from the outside to the axis, owing to the temperature fluid metal is fed constantly from the sides of the "bulb," the whole process moving progressively down the axis of the cone. The velocity of the jet with

13

14



FIGS 13 AND 14—MICROSTRUCTURES NEAR BOTTOM OF SLUG. × 500.
Fig 13, recrystallized ferrite and recrystallized pearlite, plus graphite.
Fig 14, recrystallized ferrite and spheroidized pearlite plus graphite.
Original magnification given. Reduced one fourth in reproduction.

gradient established by explosive pressure. The "orifice" consists of a partially constricted channel made by the flow of metal toward the axis of the cone, the metal having reached a temperature and state of fluidity just below that of the metal in the "bulb." The reservoir of respect to a fixed point would then be equal to the velocity with which the metal is fed into the "bulb" plus the velocity of the point of contact of the collapsing walls.

From results of experimentation, it appears there is a relationship between

the cavity-liner wall thickness and the momentum of the jet. As wall thickness is increased from zero, the penetration effect of the jet increases also until an optimum thickness is reached, after which the penetration of the jet decreases as the walls become thicker. Then, based upon the conception of collapse described above, the property of cohesiveness of the metal also seems to be an important factor in determining the suitability of a metal for cavity liners.

When the cavity walls are too thin, apparently they do not possess the necessary mass to form both an "orifice" and a "bulb" and to furnish a quantity of plastic material to form the jet. Hence, jets from thin liners possess relatively low momentum. On the other hand, when the wall is thicker than optimum it offers too much resistance to the transmission of the energy of the explosion. As wall thickness increases above the optimum relatively less and less material goes into the jet, and if it is increased until the entire cavity is filled with metal the total mass would move forward with a velocity lower than the velocity of detonation.

The optimum thickness of a conical cavity liner probably would also vary according to the physical and mechanical properties of the metal employed.

Theory of Jet Formation for Hemispheres

The mechanism of jet formation in hemispherical cavities is not as clearly defined as it is in conical cavities. Slugs from hemispherical charges have not been recovered in complete enough form to enable metallographic studies to be carried out on them. If hemispherical cavity liners are regarded in the same manner as conical cavity liners with reference to the ratio of mass of the jet to the mass of the liner, and the accompanying relationships between pressure, velocity, and energy, the same theory should apply in both cases.

In the case of cones it appears certain that the particles that form the jet are ejected by compressional forces. The same probably is equally true of hemispheres. At the time when the detonation wave strikes a hemisphere, the inside layers at the apex are subject to compressional forces. As collapse proceeds, the region of compressional forces moves down the inside of the liner until collapse is complete (Fig 2). If the liner turns inside out the portions of the metal that were subject to compression are in turn subject to tensional forces. It is from the regions of compression that the jet particles must emerge. The region of compression would be composed of a symmetrical section of the hemisphere, beginning as a point and widening into a larger and larger circle as the process of collapse advances. The particles would then be ejected normal to the surface, which would focus them along the axis of the cavity to form the jet.

Apparently the particles thus squeezed out move parallel to the axis of the hemisphere. This is assumed for the following reasons: Cast-iron hemispheres[2] make holes of relatively small diameters in steel plates, while more ductile metals make funnel-shaped holes with a diameter at the first plate almost equal to the diameter of the hemisphere. This would lead to the belief that with cast-iron hemispheres the process of collapse is interrupted if not terminated by rupture of the liner while collapse is only partially complete. For more ductile metals the process of collapse probably continues down the entire height of the liner, the lower portions of the liner provide a wide jet, which has less penetrating force per unit area and only acts to widen the hole already produced by the portion of the jet emanating from the upper portion of the cavity liner. From the present

concepts it appears that jet impact effects for a given explosive vary over a small range for corresponding relatively large changes in the apex angle in cones. Jets from acute-angle cones may penetrate deeper because of the greater length (or the greater period of duration) of the jet. It is believed, however, that increased penetration is not caused only by increased jet velocity due to a change in the apex angle of the cone but that other factors enter in as well.

Explosive Force and Structure of Metal in Liner

A description of the metallographic changes in the microstructure of a cast-iron liner from a 6-in. shaped charge has been given. The time required for the complete deformation of the cone was in the neighborhood of 10 or 15 microseconds, taking the velocity of detonation of 100 pct blasting gelatin as 26,200 ft per sec. This ultra-high-speed deformation involving high pressures applied at velocities that are extremely high in comparison with metal deformation under ordinary conditions had the effect of making the cast iron ductile enough to flow quite readily.

The essential changes that have taken place in the iron of the slug from surface to axis range from a limited amount of deformation at the surface through a phase of more complete deformation, recrystallization of ferrite and agglomeration of graphite to an area of partial spheroidization in the region of the axis of the slug.

Considering the microstructure of the slug in terms of the mechanics of jet formation, there are a number of elements that may be given consideration. The forces acting on the liner during the explosion are represented by vectors of equal magnitude in Fig 3. The only upward force of any magnitude would be one due to the inertia of the slug itself. Hence, the forces involved are mostly compressional forces or those of shear. Microscopically, flow structure varies in character from irregular multidirectional undisturbed patterns to relatively straight lines in regions of shear and greatest flow. Fig 8 represents a region of differential flow; that is, the ferrite and the graphite in the right two thirds of the photograph have been subject to flow while the pearlite in the upper left seems relatively undisturbed.

In addition, there is an almost complete welding of the grains of the cast iron along the axis of the slug. At the axis, too, the particles of ferrite and the pearlite colonies are very fine, portions of the ferrite being recrystallized. The grain size of ferrite increases toward the outside of the slug and the amount of recrystallization of the ferrite decreases.

Well-developed shear planes are shown in Figs 9 and 11, the upper section in each case showing greater relative motion toward the axis of the slug. This is clearly indicated by the drag on the lower block of metal at the lower left side. The process of collapse apparently involved the movement of rigid colonies of pearlite in a "viscous medium" of ferrite and graphite until portions of this flowing heterogeneous mass reached the vicinity of the axis, where it was probably pulverized or partially spheroidized to form a Munroe jet. The fine-grained iron in Figs 12 and 13 undoubtedly represents the approximate type of material that goes to form a jet. This would be composed, then, of a very high-velocity spray of fine graphite and fine particles of ferrite and pearlite.

Summary and Conclusions

1. A theory based upon the microstructure of a collapsed cast-iron cone is offered to explain some of the physical laws involved in jet formation.

2. A study of the microstructure of a collapsed cavity liner shows that cast iron

is ductile under conditions of temperature and pressure involved and indicates that jets from cast-iron liners are composed of a fine spray of graphite, partially spheroidized pearlite and ferrite.

3. Collapse of cast-iron conical liners appears to involve processes of compression, shear, and extrusion of particles from the inside of the liner.

4. Jet formation from cones also involves flow of metal subject to intense pressures and temperatures that are below the melting point of the metal. The metal appears to obey the laws of hydraulic flow.

ACKNOWLEDGMENT

The section on the metallographic survey of the cast-iron slug was written by W. H. Bruckner.

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Diesel Power for Underground Haulage

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(Denver Meeting, October 1947)

Introduction

Probably no other type of equipment is now being introduced into American underground mines about which less is known and about which there is more misinformation than the Diesel mine locomotive. Any equipment installed underground brings with it hazards that were not present before, and it is the purpose of this paper to show that Diesel-powered equipment can be used with safety in underground mining if certain feasible but very important precautions are taken.

Diesel locomotives are not new in underground mining, since they have been used successfully in European mines for at least 19 years, but their use in American mines is exceedingly limited; so far as is known, only five Diesel mine locomotives are operating at present in United States mines. Diesel locomotives were permitted underground in Great Britain for the first time in 1939, and the rapid increase in the number in use in those mines testifies to their safety and efficiency. More than 1300 Diesel mine locomotives are in use in European mines, according to the latest statistics; these locomotives are employed in coal, iron, lead-zinc and mercury mines. So far as is known, the Diesel mine locomotive has not yet caused an explosion or mine fire, and has not been the cause of any fatalities from toxic gases in the mines in which it has been used.

Published by permission of the Director, Bureau of Mines, U. S. Department of the Interior. Manuscript received at the office of the Institute Nov. 1, 1947. Issued as TP 2384 in Mining Technology, July 1948.

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Adjustments of Diesel Engines for Underground Use

A Diesel engine adjusted to work in an underground mine should not be confused with the same engine adjusted for use outside the mine. Adjustment of the air:fuel ratio for engines that are used on the surface is intended to give the maximum fuel economy, and little or no attention is given to the chemical composition of the exhaust gas produced. Many such engines are poorly adjusted and perhaps poorly maintained, with the result that the engine emits smoke in quantity as well as the disagreeable odor familiarly associated with Diesel engines. The air:fuel ratio in the case of Diesel engines operated on the surface is generally around 15:1; that is, 15 lb of air to burn 1 lb of fuel.

Diesel engines adjusted to work underground usually have an air:fuel ratio of about 20:1, as at this ratio the minimum quantity of toxic gases is given off in the engine exhaust. Diesel engines operated underground must be maintained in better condition than similar engines used on the surface; otherwise, the smoke emitted in the exhaust becomes objectionable if not unendurable.

Practically normal air is required for safe operation of Diesel engines underground or in confined places; failure to provide virtually normal air for Diesel engine operation is almost certain to result in trouble. Bureau of Mines engineers have determined that the air in which a Diesel operates in confined places should contain less than ½ pct of carbon dioxide and at least 20 pct of

oxygen. In a test, a Diesel locomotive was run in a confined place until the carbon dioxide concentration in the atmosphere was approximately 1 pct and the carbon monoxide was 0.01 pct; visibility was blanked out, and the odors were sickening.

Hazards Attending Use of Diesel Engines Underground

It is important to recognize the hazards attending the use of Diesel-powered equipment underground and to know how these hazards are overcome in practical mine operations.

Toxic Constituents of Exhaust Gas

The toxic constituents of Diesel-engine exhaust gas are carbon dioxide, carbon monoxide, and oxides of nitrogen. The characteristics of the first two are fairly well known to those connected with the mining industry, but the dangerous effects of breathing oxides of nitrogen are not well known, although these are the most dangerous of the toxic gases in the exhaust.

Carbon dioxide is an odorless, colorless gas, heavier than air and may accumulate near the floor of a mine working improperly ventilated. Carbon dioxide is formed by complete oxidation of the carbon in the fuel oil. One-half of 1 pct of carbon dioxide in normal air causes a slight increase in the lung ventilation. If there is 2 pct of carbon dioxide in the air, the lung ventilation will be increased about 50 pct. Air containing 10 pct carbon dioxide cannot be endured more than a few minutes.

Carbon monoxide is also an odorless, colorless gas and is slightly lighter than air. It diffuses easily into the surrounding atmosphere but may accumulate in or toward the top of a working place that is not well ventilated. Carbon monoxide in the exhaust gas is formed by incomplete oxidation of the fuel oil; contrary to the belief of many people, carbon monoxide is always present in the exhaust of a Diesel engine. It is extremely hazardous to breathe, even in small amounts. Carbon monoxide in excess of 0.01 pct eventually may produce symptoms of poisoning, and 0.02 pct will produce slight symptoms in several hours. When 0.04 pct is present and the exposure is for 2 or 3 hr, headache and discomfort usually occur. With moderate exercise 0.12 pct will produce slight palpitation of the heart in 30 min., a tendency to stagger in 1½ hr, and confusion of mind, headache and nausea in 2 hr. A concentration of 0.20 to 0.25 pct will usually produce unconsciousness in about 30 min.

"*Oxides of nitrogen*" is the term applied to the oxides of nitrogen or mixtures of them found in the exhaust of the Diesel. The most serious consequence of inhaling these gases is the formation of nitric and nitrous acids in the respiratory system. The maximum amount of this gas considered permissible in a working environment is only 25 ppm, or 0.0025 pct. The oxides of nitrogen in the exhaust are formed solely by the reaction between the oxygen and nitrogen composing the air when the air is heated to a very high temperature by the pressure inside the cylinder of the Diesel engine. The fuel oil does not enter into this chemical reaction. One one-hundredth (0.01) of 1 pct of nitrogen peroxide may cause serious illness if breathed for a short time (½ to 1 hr) and 0.02 to 0.07 of 1 pct is fatal if breathed for a short time.

The relative order of toxicity of oxides of nitrogen is shown by comparison with carbon monoxide. An atmosphere containing 0.5 to 1.0 pct of carbon monoxide is likely to be fatal to a human being in less than 30 min. of exposure, whereas as little as 0.07 pct of oxides of nitrogen may be fatal in the same time.

Aldehydes cause the odor associated with the exhaust of a Diesel engine; it is the only odor that can be detected in the exhaust gas because the other gases are odorless,

except the slight traces of oxides of sulphur that may be present. The odor of aldehydes resembles that of formaldehyde, which is the aldehyde of formic acid. Aldehydes are so irritating to breathe that it is unlikely that men could continue to work in an atmosphere containing dangerous amounts. The maximum concentration in air to be breathed considered permissible by the U. S. Public Health Service and many State Health Departments is 10 ppm.

Prevention of Toxic Gases

Mention of the toxic constituents of Diesel-engine exhaust gas and their characteristics has been made to call attention to this hazard in connection with the underground use of the Diesel engine. The prevention of danger from this source is simple—the engines should be well designed, should be kept in good repair, and enough ventilating air should be provided to dilute and carry away the toxic gases from the engine exhaust. The minimum volume of ventilating air required for operating Diesel engines underground is about 75 cu ft per minute per brake horsepower; this volume of air is required for each Diesel engine in service in any particular place and is additional to the volume of air required for normal ventilation, insofar as quality is concerned. However, if an excess of ventilating air is supplied for other reasons, it is possible that no additional air will be required if a Diesel engine is placed in operation. Take, for example, in addition to the minimum quantity of air required to ventilate a nongassy coal mine, if two Diesel 65 bhp mine locomotives are placed in service, then additional ventilating air must be supplied as follows:

$$2 \times 65 \times 75 \text{ cfm} = 9{,}750 \text{ cfm}.$$

This would be the additional amount of ventilating air that would need to be supplied to maintain the mine atmosphere at the proper quality level.

However, if at this same mine it is necessary to supply 100,000 cu ft of air per minute to keep the methane content of the mine air within the 0.25-pct limit, no additional air would be required to operate the two Diesel locomotives.

Washers or scrubbers can remove most of the aldehydes from the exhaust gas. A washer consists essentially of a container or tank of water through which the exhaust gas passes; baffles are used to assist in breaking the bubbles of gas. Water is one of the best absorbents of aldehydes, but considerable care should be taken to keep the proper amount of water in the washer and to change the water before it becomes saturated with aldehydes and can no longer function. A washer does not remove the carbon monoxide from the exhaust gas but does remove much of the aldehydes and some of the oxides of nitrogen. A properly designed washer that is well maintained will remove most of the typical Diesel odor and usually decrease employee opposition to the use of Diesel-powered equipment underground.

Flame arresters on both the intake and exhaust of Diesel engines used in coal mines is a "must" because of the ignition hazard if an explosive mixture of gas and air is encountered. In metal mining, the use of flame arresters is not as important unless there is a possibility that an explosive gas may be present, as in some tunnels and mines. A flame arrester on the exhaust is desirable in any type of mine as it minimizes the fire hazard.

The mining laws and regulations of some states prohibit the use of internal-combustion engines in underground mines; other states have laws and regulations permitting their use providing permission is obtained from the State Department of Mines before installation. The laws and regulations prohibiting the underground use of internal-combustion engines are intended primarily to prevent the use of gasoline-powered mine locomotives and

other equipment. These laws and regulations were placed in effect before the Diesel engine was improved to its present stage of safety and efficiency, and it now appears that alterations should be made in state laws and regulations to allow the use of Diesel equipment under certain prescribed conditions.

Chemical Composition of Exhaust Gases

It is important to understand the difference in the chemical composition of the exhaust gas of a Diesel engine and a gasoline engine insofar as the toxic gases are concerned. The carbon monoxide content of the actual exhaust gas of a gasoline engine may amount to as high as 14 pct, while the carbon monoxide in the exhaust of a Diesel engine adjusted for underground use and in good mechanical condition will range from 0.02 to 0.05 pct. However, if the Diesel engine is not in proper adjustment for underground service and is not in good mechanical condition, the exhaust of the Diesel can be very hazardous, virtually as serious as that of a gasoline engine.

The exhaust gas of a Diesel engine when properly adjusted contains an excess of oxygen, even when the engine is working under full load and at maximum speed. Typical exhaust-gas analyses when the engine is working under full load and when idling are shown in Table 1.

Table 1—*Typical Diesel-engine Exhaust-gas Analyses*

	Per Cent	
	Engine Idling	Engine Under Full Load
Oxygen	19.63	14.98
Nitrogen	79.40	80.63
Carbon dioxide	0.89	4.31
Carbon monoxide	0.03	0.05
Methane	0.05	0.03
	100.00	100.00
Oxides of nitrogen, ppm	20	43
Aldehydes, ppm	40	

Underground Operations

A Limestone Mine in Ohio

The limestone mine in Ohio from which the exhaust samples of Table 1 were taken is opened by two shafts about 2200 ft deep; the two-entry, room-and-pillar method of mining is used. The entries are driven about 16 ft wide and 17 ft high; the rooms are driven approximately 30 ft wide and 17 ft high for 600 ft. On retreat an additional 12 ft of top rock is mined, making the final room size about 30 ft wide and 29 ft high; the pillars are not recovered. The mine is operated on two shifts per day, and the daily production averages 1800 tons of limestone. Approximately 70 men are employed underground on each shift.

Ventilation is induced by a reversible fan operated blowing. Approximately 66,000 cu ft of air per minute is coursed through the mine in two splits; the haulageways are ventilated by return air when it is possible to do so.

The Diesel equipment used underground at this mine includes six 6-cu yd-capacity trucks powered by 107-hp engines, a bulldozer, a tractor loader and a service truck. Analyses of the samples of the general mine air taken at two working faces and in the main return at the shaft bottom where three Diesel trucks had passed in the preceding 7 min. were as given in Table 2.

Table 2—*Analyses of Samples of General Mine Air*

	Per Cent		
	Face of Room A	Face of Room B	Main Return at Shaft Bottom
Oxygen	20.92	20.72	20.92
Nitrogen	79.02	79.08	79.02
Carbon dioxide	0.06	0.16	0.05
Carbon monoxide	0.00	0.00	0.00
Methane	0.00	0.04	0.03
	100.00	100.00	100.00

The successful use of Diesel-powered equipment at this mine is significant. The

mine shaft is about 2200 ft deep, and ventilation of the working places is induced by mechanical methods. The volume of fresh air taken into the mine is adequate to dilute and carry away the toxic gases of the exhaust from the Diesel engines, and analyses of the general mine air indicate that the air in the working parts of the mine is better than ordinarily found in mines, notwithstanding the use of seven, and sometimes nine, units of Diesel-powered equipment.

In the Central States, Diesel-powered trucks have been used for several years as haulage units in some of the underground limestone mines having adit openings; more recently, other Diesel-powered equipment, such as bulldozers, portable air compressors and tractor-mounted drill jumbos, has been used with success.

A Limestone and Shale Mine in Missouri

An underground limestone and shale mine in Missouri is using internal-combustion engines as power units for most of the underground mining operations other than loading rock. Portable air compressors of 350 to 500 cfm capacity are powered by Diesel engines; rock-drill jumbos are mounted on Diesel-powered caterpillar tractors and moved from one working place to another as required. In moving, the Diesel tractor pulls the portable compressors to the new working place; the compressor is left a convenient distance from the face, generally in a crosscut between two pillars, and the jumbo is taken to the drilling position.

In this mine, the limestone is virtually level and is overlain with approximately 100 ft of cover. The mine is opened by two adits, and the mined-out area covers approximately 80 acres. The mining method consists of driving rooms about 38 ft wide between staggered pillars, roughly 55 ft square. The rooms are driven about 18 ft high, and all mining in limestone is advancing. The shale that overlies the limestone is mined by blasting about 18 ft of the overhead shale into the rooms, where the limestone mining is completed or abandoned temporarily. Shale mining is worked on the retreat; the mine production averages approximately 800 tons daily. The rock is loaded by an electric shovel into trucks of 15-cu yd capacity; three trucks are powered with gasoline engines and one truck with a Diesel engine. The rock is hauled to the crusher, which is a short distance from the portal of the adit.

Mechanical ventilation is not provided, but the natural ventilation is assisted with two vertical, concrete-lined circular shafts; one shaft is 14 ft and the other 7 ft in diameter. Considerable air movement in the mine is induced by the piston effect of the large trucks moving through the relatively narrow haulageways. Air measurements indicate that approximately 20,000 cu ft of air per minute is circulating through the working sections of the mine.

The general mine air along the haulageways and in the vicinity of the working faces was sampled with vacuum tubes and by using a supersensitive carbon monoxide indicator and is given in Table 3. The carbon monoxide concentration was not sufficient to be shown on the indicator, which accurately detects carbon monoxide concentrations in the range of 0.1 to 0.001 pct and 0.0025 pct can be estimated.

TABLE 3—*Samples of the General Mine Air*

	Per Cent		
	About 150 ft from 350-cfm Portable Air Compressor	Near 500-cfm Portable Air Compressor	Face of Room Electric Shovel Was Operating
Oxygen	20.76	20.81	20.87
Nitrogen	79.08	79.07	70.03
Carbon dioxide	0.01	0.11	0.09
Carbon monoxide	0.00	Less than 0.01	Less than 0.01
	100.00	100.00	100.00
Oxides of nitrogen, ppm	12		

The general mine atmosphere is virtually normal air and in quality is better than is usually found in underground mines. The use of gasoline engines on the three trucks is regarded as hazardous by the Bureau of Mines, not only because of the carbon monoxide in the exhaust; but, in addition, the explosion hazard of gasoline is much greater than when the less flammable fuel oil is used in Diesel engines.

A Lead-zinc Mine in Colorado

A Diesel mine locomotive has been in service as a mainline haulage unit at a lead-zinc mine in Colorado for over 2 years and is reported to be satisfactory as to both safety and efficiency. The locomotive weighs 6 tons as installed and is powered with a Diesel engine rated as 65 bhp at sea level; because of the high altitude of the mine (10,600 ft) the engine delivers only 55 bhp at the mine.

The locomotive pushes 15 empty 3-ton-capacity cars to an inside yard about 8500 ft from the tunnel portal and takes 15 loaded cars to the mill, a distance of about 2 miles. The capacity of the engine is estimated to be 700 ton-miles, divided into 500 ton-miles of loaded cars and 200 ton-miles of empty cars.

The Diesel engine is equipped with a water-type exhaust-gas washer that appears to be efficient; the odor of aldehydes was not perceptible along the haulageways or in the working sections of the mine.

Ventilation was by means of a suction fan at the surface and an 18-in. steel pipe; a booster fan was installed 8500 ft from the portal, and it is stated that 5000 cu ft of air per minute was delivered to the mine workings.

Analyses of samples of the mine air taken at the end of the Diesel run (8500 feet from the portal) are shown in Table 4. These samples probably are not indicative of the quality of the mine air at present because, since these samples were taken, a connecting raise has been completed to the mine workings above the tunnel.

TABLE 4—*Analyses of Samples of Mine Air*

Sample number	1	2	3
Date of sampling	7/12/46	7/12/46	9/5/46
Oxygen, pct	20.25	20.12	19.98
Nitrogen, pct	79.34	79.41	79.56
Carbon dioxide, pct	0.41	0.47	0.46
Carbon monoxide, pct	0.00	0.00	Less than 0.005
	100.00	100.00	100.00
Oxides of nitrogen, ppm	17	13	28

The use of Diesel mine locomotives at such a high altitude (10,600 ft) is unusual; most European mines using Diesel locomotives underground have an altitude of less than 1500 ft. A mine in South America situated at approximately the same altitude (10,000 ft) is reported to be experimenting with superchargers on the intake of a Diesel engine to increase the temperature inside the cylinder walls caused by pressure.

A Railroad Tunnel

A railroad tunnel 2660 ft in length is being driven in one of the eastern states, and internal-combustion engines are used as power units. The equipment consists of:

1—Diesel-powered shovel (113 bhp).
3—Diesel-powered trucks (190 bhp) with 10 cu yd capacity.
1—Diesel-powered bulldozer (113 bhp).
1—Explosives delivery truck powered with gasoline engine.
1—Small bulldozer (spare) powered with gasoline engine.
1—Truck for general hauling, powered with gasoline engine.

A pilot tunnel approximately 15 by 15 ft in section was driven for the entire length before the main tunnel was started. The center lines of the pilot tunnel and the main bore coincide; the main bore dimensions in the rough are approximately 36 ft in width and 34 ft in height, with an arched

roof making the cross-sectional area about 1000 sq ft. There is a difference of 8 ft in elevation of the portals.

The tunnel is ventilated with a centrifugal blower situated just outside the tunnel portal; a 30-in.-diam steel pipe extends to within about 50 ft from the face. The blower is rated at 12,000 cfm, but while exhausting the air quantity as measured was 13,600 cfm. There is some natural ventilation through the pilot tunnel at most times. The blower is arranged so that it may be operated either to supply exhaust or pressure ventilation to the tunnel. After blasting, the blower is operated as an exhauster for 20 to 30 min., then as a blower for about 10 min., and again as an exhauster during the whole mucking operation. Sometimes the air conditions at the face appear to be better when the blower is operated as a blowing unit, in which case this is continued while mucking.

The mucking-cycle procedures are as follows: The three Diesel trucks and the Diesel bulldozer are outside the tunnel during blasting operations, and the Diesel shovel is moved back a safe distance from the face. After blasting and removal of the smoke from the explosives, the bulldozer and trucks enter the tunnel. The bulldozer proceeds to the face and pushes the scattered rock from blasting to the foot of the muck pile and then leaves the tunnel. The Diesel shovel moves into loading position, and the trucks advance in turn for loading; the truck engines are stopped when the trucks are not in motion. The Diesel engine on the shovel operates continuously during the loading operation; the loading requires about 3 hr. Ten trucks were loaded during the first hour of the observed mucking operation.

Analyses of air samples taken during mucking operations are given in Table 5.

The analysis of the air sample taken at the start of the mucking operation indicates that the carbon dioxide and carbon monoxide present are caused by the gases remaining in the face area after blasting as the Diesel equipment had just started to work. It is evident that the quality of the air in the tunnel is good; the large cross-sectional area of the tunnel tends to minimize any contamination of the mine air caused by Diesel-engine operation.

Table 5—*Analyses of Air Samples taken During Mucking Operations*

	Per Cent		
	60 Ft from Face at Start of Mucking Operations	60 Ft from Face after 1 hr 20 min. of Mucking	200 Ft from Face at Truck Turn Point
Oxygen	20.78	20.70	20.91
Nitrogen	79.07	79.09	79.03
Carbon dioxide	0.15	0.21	0.06
Carbon monoxide	Less than 0.005	Less than 0.005	Less than 0.003
	100.00	100.00	100.00
Oxides of nitrogen, ppm	0	0	0

Summary

Diesel engines of suitable construction can be used safely in underground mines if adequate ventilation is provided and the Diesel equipment kept in reasonably good repair. The ventilation should be induced mechanically so the circulation of the air can be controlled at all times. The atmosphere in which a Diesel engine works should be essentially normal air, as otherwise trouble is virtually certain to be experienced.

Mine officials who are responsible for health and safety of the workers in the mine should have a thorough understanding of the hazards introduced by using Diesel equipment underground. The plan of ventilation should be worked out before the units are installed, and samples of the mine air should be analyzed at frequent intervals after the Diesels are first installed underground; the object of this is to check the effectiveness of the ventilation. Pe-

riodic analyses of the mine air, particularly along the haulageways should be made while the Diesels continue in operation; this is to check the mechanical condition of the Diesel engines as well as on the ventilation.

The Diesel mine locomotive is less hazardous in underground operation than one of the electric-trolley type. A study of American coal-mine statistics reveals that 783 fatalities were caused by electric-trolley systems in underground mines in the period 1930 to 1944,[a] while European statistics show no fatalities caused by the use of Diesel locomotives in underground coal mines.

[a] Statistics from the Accident Statistics Division, Bureau of Mines and published in *I.C.* 7328 and 7406 of the Bureau of Mines.

The Diesel mine locomotive is preferable to the storage-battery locomotive when heavy loads and steep grades are considered. The original cost of the equipment and the cost of maintenance definitely favor the Diesel insofar as now known.

For coal mines a "permissible" Diesel engine should be used because of the danger of possible ignitions of gas-air or coal-dust mixtures by flame from the exhaust. Up to the present time, no "permissible" Diesel mine locomotives are on the market.

The Diesel engine should not be used in any type of an underground mine or in any confined place unless adequate ventilation is provided. If this is done, the Diesel engine can be adopted with safety in underground mines.

Experiences with Acid Mine-water Drainage in Tri-State Field

By O. W. Bilharz,* Member AIME

(New York Meeting, March 1947)

Introduction

Acid mine-water drainage is a serious problem with many mines in the Tri-State zinc and lead mining district. Particularly is this true when large volumes must be considered in unwatering old mines that have refilled with water which has dissolved the products of oxidation and has become chalybeate in character.

The experiences discussed in this paper have to do specifically with the unwatering of a comparatively small area west of the town of Baxter Springs, Kans., where the pumping job was complicated by water of this type and where the extremely high sulphate content particularly effected that phase of mine drainage having to do with stream pollution.

Location, Geology and History

The Baxter Springs area of the Tri-State mining field is situated just west of the town of Baxter Springs and is a northeasterly extension of the main mineralized zones of the Oklahoma part of the district (Fig 1).

The mining area is traversed with well defined shear zones which provide easy circulation of ground waters and the ore bodies are found along these shear zones as localized enrichments wherever the structural conditions are favorable (Fig 2). The ore-bearing strata is Mississippian with an overlain capping of shale of Pennsylvanian age. This shale strata carries much pyrite and marcasite at its base and undoubtedly was a contributing factor to the subsequent high iron content of the water.

The area originally contained a sulphide water which discharged as an artesian flow in springs within the town and gave Baxter Springs its name. These springs were famed for many years for their mineral water.

This original sulphide water was first removed about 1917 and active mining was carried on west of the town during subsequent high metal price periods. Many small mines were opened, with each localized enrichment becoming a separate mine. During the depression mining ceased and all mines in the west Baxter area were allowed to flood. No serious effort was made to again unwater until higher prices caused from the demand for metal for World War II once more focused attention on this part of the field.

Original Mining Results in Oxidation

As plans for unwatering progressed, it soon developed that the mine-water pools had become strongly acid in character and carried large amounts of sulphates in solution. This occurred in the following way. When the original sulphide water was removed, shafts sunk, and mining carried on over a number of years, ferrous sulphate was formed by the spontaneous oxidation of the pyrite beds in contact with air admitted through churn-drill holes and mine openings. These products of oxidation can be seen today in the old mines of the district as crusts of crystalline sulphates

Manuscript received at the office of the Institute March 24, 1947. Issued as TP 2267 in Mining Technology, November 1947.

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on the walls of stopes and as stalagtites and stalagmites formed under roof drips. Ferrous sulphate predominated, but large amounts of other sulphates were precipi- period of mining. With the cessation of mining operations during the depression the entire area refilled with water and the accumulated sulphates formed by this oxi-

Fig 1—Tri-State Kansas-Oklahoma area.

tated as oxidation products of the alkaline carbonates and bicarbonates. The oxidation covered a large area because of the fact that the shear zones were very open and permitted easy circulation of air during the dation were dissolved. As the water table rose, a large amount of ferric sulphate was also formed from the oxidation of the ferrous sulphate together with the basic ferric sulphate.

$$6FeSO_4 + 3O + 3H_2O \rightleftarrows 2Fe_3(SO_4)_3(OH)_3 \rightleftarrows 2Fe_2(SO_4)_3 + 2Fe(OH)_3.$$

The action of this ferric sulphate took iron into further solution from the pyrite when contact was made with the pyrite beds, and more ferrous sulphate was formed.

$$Fe_2(SO_4)_3 + FeS_2 \rightleftarrows 3FeSO_4 + 2S$$
$$2S + 6Fe_2(SO_4)_3 + 8H_2O \rightleftarrows 12FeSO_4 + 8H_2SO_4$$
$$Fe_2(SO_4)_3 + H_2O \rightleftarrows 2FeSO_4 + H_2SO_4 + O$$
$$Fe_2(SO_4)_3 + Fe \rightarrow 3FeSO_4$$

Since iron predominated, only a few of the iron reactions are given, but similar reaction formulas can be written for most of the other heavy metals.

Theory of Electrolytic Dissociation Applicable

According to the theory of electrolytic dissociation, water, or any liquid containing water in good measure, contains free positively charged hydrogen or (H) ions and negatively charged hydroxyl (OH) ions. When these are in balance, the solution is said to be neutral and have a pH value of 7.0; i.e. the equality of the (H) ion and the (OH) ion concentration occurs at pH 7.0. Decinormal hydrochloric acid has a pH of 1.04, is completely ionized and contains one decigram of (H) ion. On the other end decinormal sodium hydroxide has a pH of 13.07.

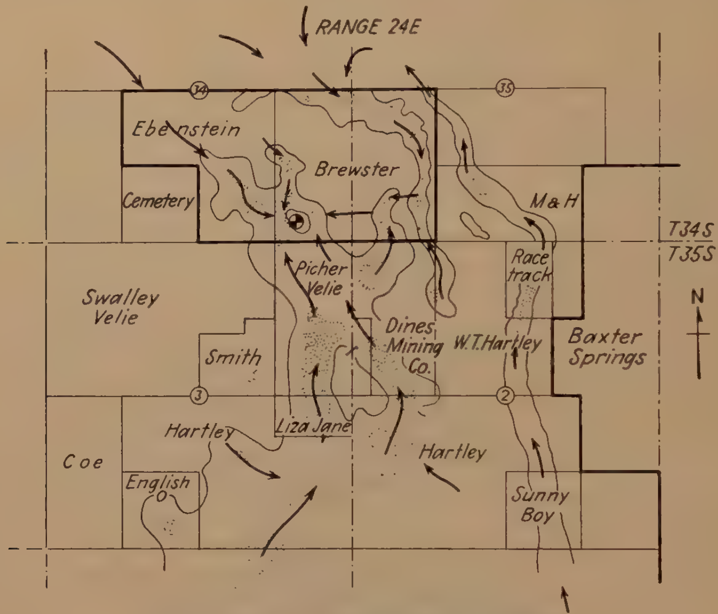

FIG 2—SHEAR ZONES SHOWING ORE BODIES.

An acid, therefore, is any substance capable of supplying to its solution hydrogen bearing a positive-electric charge and the expression of the intensity factor is the hydrogen-ion concentration, or pH. Thus the acidity of a natural water represents the content of free carbon dioxide, mineral acids, and salts, especially sulphates of iron and aluminum, which hydrolyze easily to given (H) ions.

Referring to Table 1 you will see that the mine-water analysis shows a high content of carbon dioxide and bicarbonates as $(HCO_3)_2$, as well as extremely high values

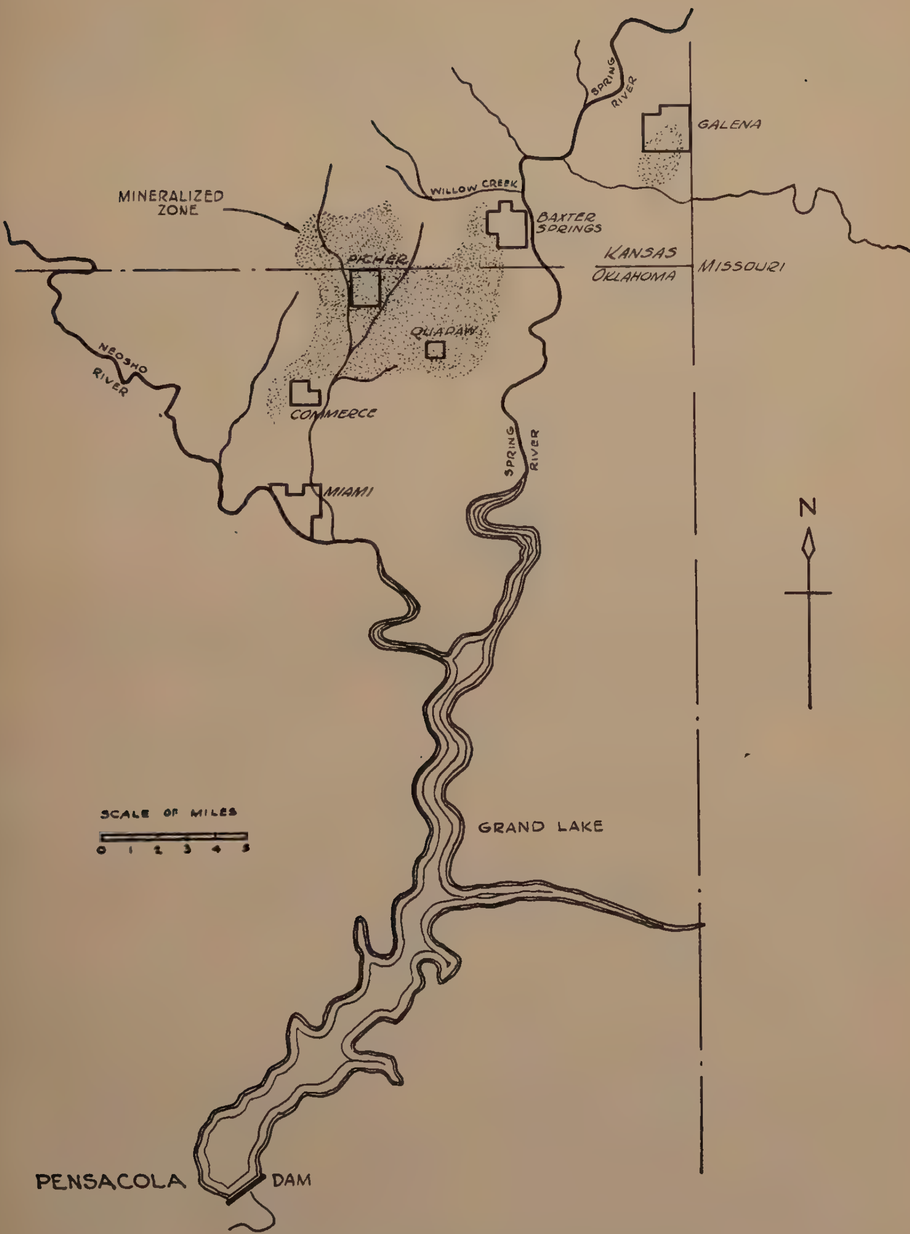

FIG 3—TRI-STATE MINING AREA DRAINAGE BASIN.

in the mineral acids and salts, particularly those of iron, which show as sulphates and all of which hydrolyze easily to positively charged (H) ions.

on these four points, therefore, our object was to discharge the mine water into Spring River so as to prevent undue change in the chemical quality of the river water.

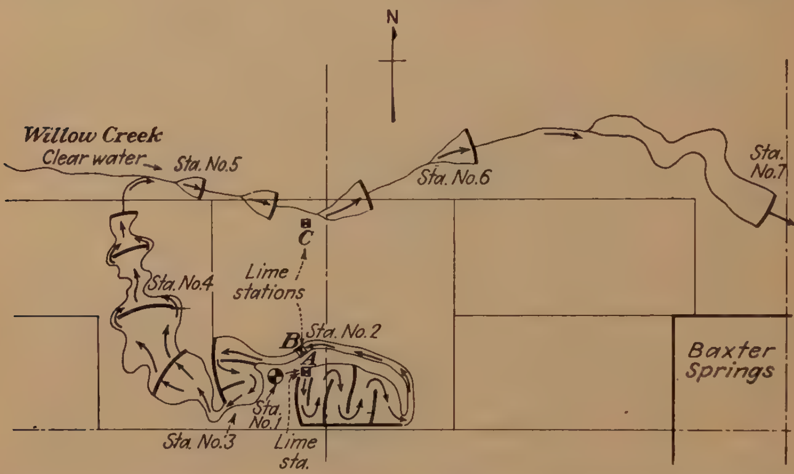

FIG 4—PLAN OF LIME-TREATMENT CIRCUIT WITH SETTLING BASINS AND LIME STATIONS.

All this spelled acid water and it is not surprising that an analysis shows the hydrogen-ion concentration at a pH of 2.45.

STREAM POLLUTION PROBLEM

It was necessary to pump a large pool of this type water in order to unwater the mining area, and it was evident that if this water was discharged into stream drainage without treatment, a serious stream pollution would be created. Not only would it have detrimental effects on the use of Spring River from a recreational standpoint but it would also bear on the fishery problems in the Pensacola Dam Impoundment of the Grand River into which drainage would flow via Spring River (Fig 3). The effect of high concentrations of this type of water would be severe on fish life because of the acidity, the toxic metals in solution, the turbidity which would be created by the precipitates and the impoverishment of the dissolved oxygen of the river waters by the oxidation of the ferric iron. Based

TABLE 1—*Assay of Mine Water*[a]

Constituents	Value	In Ppm
Ferrous iron		6,510
Ferric iron		265
Calcium		586
Magnesium		663
Sodium		526
Aluminum		1,029
Zinc		2,100
Cadmium		15
Copper		6
Lead		2
Manganese		12
Titanium		5
Phosphorus		3
Chlorine		15
Sulphates as (SO_4)		24,649
CO_2 and bicarbonates as $(HCO_3)_2$		2,637
Silica in solution		17
pH	2.45	

[a] Bruce Williams Laboratory.

PRELIMINARY STUDIES AND EXPERIMENTS

Experiments were conducted to determine what amounts and what compositions of mine water could be discharged into Spring River so that the mixture would not be harmful to fish life. Since the Pensacola Dam was in process of completion, particular concern was voiced by the Oklahoma Fish and Game Department

over the possibility that mine-field wastes, entering the Impoundment via Spring River, might constitute the greatest single potential pollution hazard to fish and other was effected with the Oklahoma Fish and Game Department. Stations were set up where daily water samples could be taken and analysis made to determine the acidity

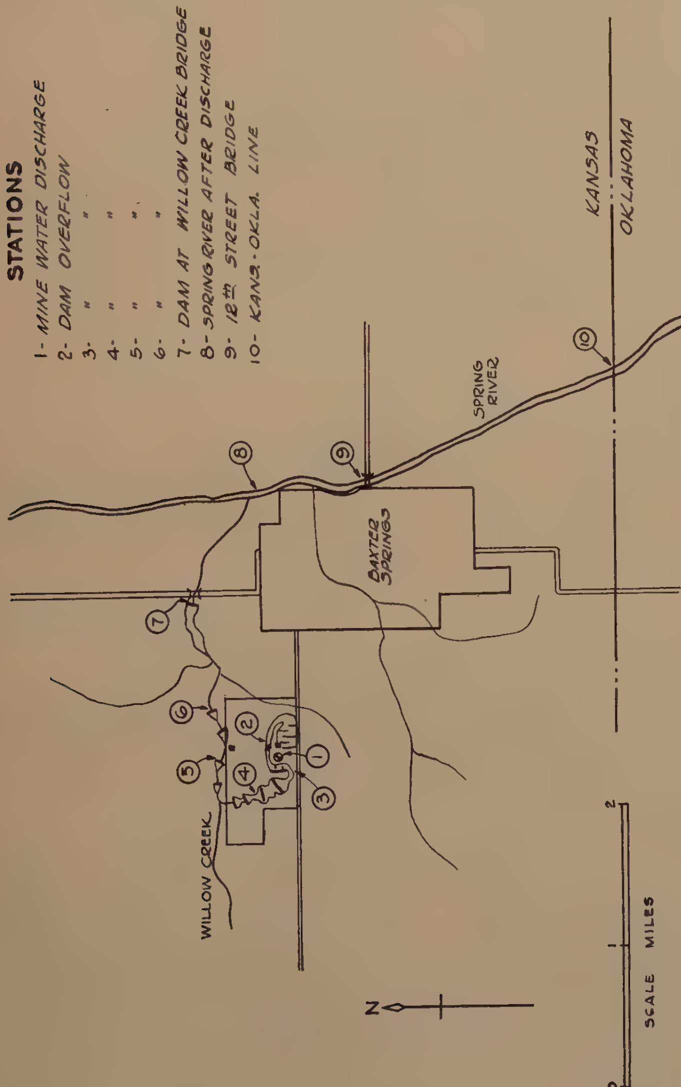

FIG 5—PLAN OF LIME-TREATMENT CIRCUIT WITH SAMPLE STATIONS AT SPRING RIVER POINTS.

aquatic life in the completed reservoir. While no definite allowable limits on stream pollution have ever been set for Oklahoma streams, a reasonable control caused by mineral acids and sulphates, alkalinity in ppm as $CaCO_3$, the chlorides in ppm as Cl, the sulphates in ppm as SO_4, and provision made to keep a close check

on the pH and iron content both as ferrous and ferric.

Lime-treatment Circuit Established

Based on the results of these experiments to control the pollutant, the decision was made to design and establish a treatment circuit for the use of calcium oxide in the form of hydrated lime and an elaborate system of treatment plants and settling basins was constructed. The first function of the process was to raise the pH value, by calcium oxide treatment and aeration,

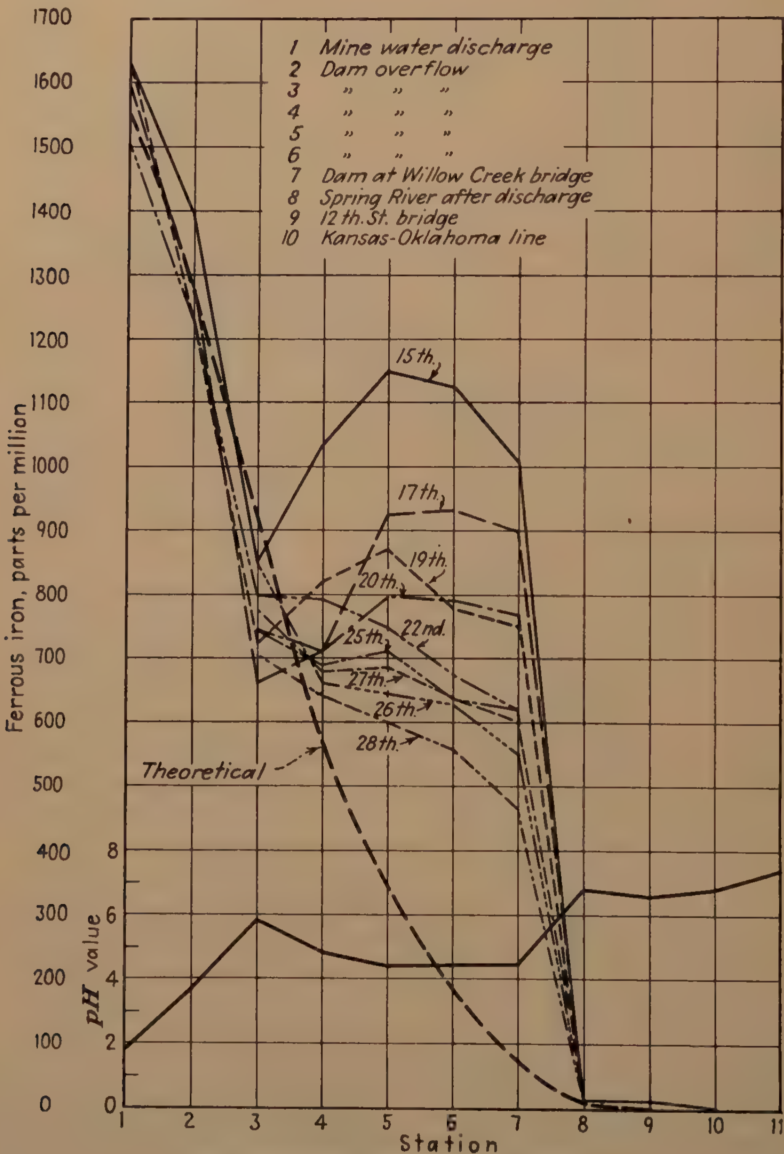

Fig 6—Ferrous iron in pond water.

to a point where precipitation of iron would take place. When the mine water was pumped to the surface it was crystal clear and was apparently saturated with iron ions and in a state of equilibrium, so that no precipitation of iron took place until sufficient oxygen had been absorbed. Considerable amounts of free carbon dioxide were also present in the water and since the iron-precipitating process involved the formation of carbon dioxide, it was necessary that provision be made for plenty of agitation and aeration to remove excessive amounts of CO_2. This was provided before and after treatment with hydrated lime at the various lime stations, and the settling basins were constructed of shallow depth to permit much contact with the air and to assure more thorough utilization of the lime and adequate precipitation and settling conditions. The system was also designed to take care of a maximum flow of water since it was desirous to unwater as speedily as possible, on the premise that mine-water inflows would be more neutral than the supersulphated pools.

Fig 4 shows how the circuit was designed to provide as much settling area as possible, and also shows the location of the liming and aeration stations. Fig 5 shows the same in smaller scale and locates the additional sample stations at Spring River points.

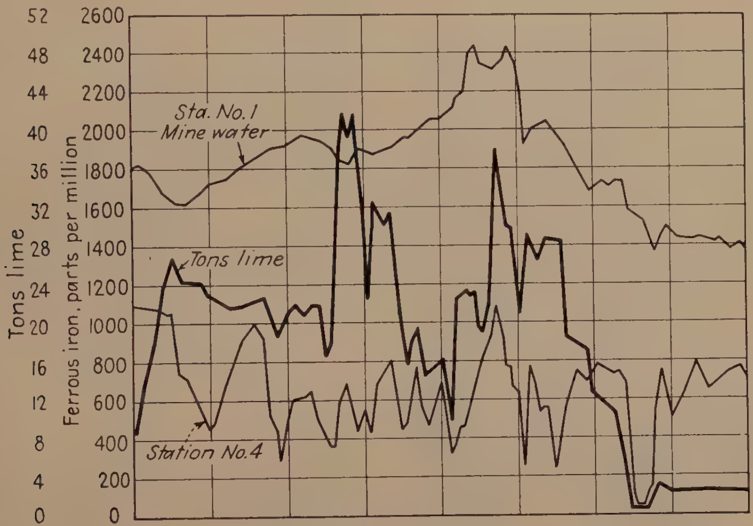

FIG 7—LIME CONSUMPTION.

Fig 6 shows graphs of the ferrous iron control when the circuit was put into operation. You will note the effect of the calcium oxide in precipitating the iron and the rise of the pH values to the point where balance was finally established. The high values following the second liming were caused by dilution of the circuit by untreated water, which occurred when a pond dam went out. This untreated water redissolved suspended iron precipitates and the full effect of the lime was not felt until the circuit was re-setablished.

EFFECTS OF DISCHARGES INTO RIVER AND RELATIVE REACTION

You will note that discharges into Spring River carried about 500 ppm ferrous iron still in solution. Experiments had shown that at a river stage of 3 ft, the approximate flow of Spring River was 44,000 gpm

and that 10 pct or about 4400 gpm of mine water running not more than 500 ppm ferrous and at a pH not lower that 4.5 would permit a dilution not injurious to fish life. The control of dilution to the limits mentioned had no effect upon the iron hydrates still left at this stage, and all the iron remaining was neutralized and precipitated at once by the excess of alkalinity and oxygen in the river and the precipitates settled out within the first mile of flow. While not injurious to fish life, this iron precipitate did prove an irritant to some landowners downstream in as much as a periodic surge of river water from a power plant upriver carried that precipitate in suspension to be redeposited in quiet water down-river where it stained shore lines with iron oxide. An additional lime station was installed in the circuit on this account and thereafter discharge in the river ran about 250 ppm ferrous.

Lime Consumption and Final Drainage Analysis

The lime consumption was calculated on the basis of pumping 2000 gpm of water carrying 1600 ppm ferrous. The hydrated lime used had a calcium oxide value of 74.20 pct with available hydrated lime at −200 mesh of 98.04 pct. Allowing for 10 pct aeration in the circuit the theoretical lime consumption was calculated at 23.6 tons per 24 hr. Control of the circuit was determined by accurate analysis and the lime additions varied up or down according to the amount of water pumped below or above 2000 gpm and the ferrous content below or above 1600 ppm. Fig 7 is a chart showing the amount of lime introduced into the circuit according to this plan. Cold weather had a retarding effect on precipitation as well as on the rate of settling of the precipitates and tended to increase lime consumption. In periods of heavy rain, however, lime consumption was lowered since it was possible to discharge more of the highly acid-mine water during flood periods when the ratio of stream to

Table 2—*Analysis of Final Drainage*[a]

Constituents	Value	In Ppm
Calcium		536
Magnesium		469
Zinc		585
Ferrous iron		2,420
Ferric iron		230
Sodium		134
Aluminum		287
Sulphates		11,005
Chlorine		25
Hydrogen		10
H in terms of free H_2SO_4		487
Carbonate alkalinity		
Bicarbonate alkalinity		
Silica		
pH	1.70	

[a] Assay of mine water by Bruce Williams Laboratory.

Table 3—*Analysis of Neutral Mine-water Inflow*[a]

Constituents	Value	In Ppm
Calcium		213
Magnesium		80
Zinc		0.07
Ferrous iron		0.08
Ferric iron		0.10
Sodium		116
Aluminum		
Sulphates		823
Chlorine		14
Hydrogen		
H in terms of free H_2SO_4		none
Carbonate alkalinity		none
Bicarbonate alkalinity		290
Silica		6
Total dissolved solids		1,542
pH	7.60	

[a] Assay of mine water by Bruce Williams Laboratory.

Table 4—*Comparative Analysis of Spring River Water*

Water Analysis	Spring River Above Mine Water Discharge	Spring River 1 Mile Below Oklahoma Line
Alkalinity, ppm as $CaCO_3$	107	74
Chlorides, ppm as Cl	9	11
Sulphates, ppm as SO_4	58.5	160
Non-carbonate hardness, ppm as $CaCO_3$	32	121
Total hardness, as $CaCO_3$	139	195
pH	7.7	7.3

mine-water volumes was large. This was particularly true during the last stages of the unwatering. All district streams were at flood stage and we were able to complete our unwatering with large volumes even though the final mine-floor drainage of the pool carried high free sulphuric acid content, as the analysis (Table 2) shows. Pumping installations were subsequently moved to deeper level mines to the south of our property and after these were unwatered the inflow into the pool, when not contaminated by roof drips, analyzed as shown in Table 3. Finally Table 4 gives an average analysis of Spring River water before mine water was discharged into it as compared with a downstream analysis of the mixture below the Oklahoma line.

Conclusions

Thus our objective, to discharge the mine water into Spring River so as to prevent undue change in the chemical quality of the river water entering Oklahoma and the Pensacola Dam Impoundment, was attained, and the unwatering of the acid-water pool accomplished.

Much has been said of the effects of acid-mine water flooding in relation to the future conservation of the Tri-State's marginal reserves. If these reserves were to be abandoned and the entire field allowed to flood, great volumes of acid water inevitably would result. When pumped in amounts necessary to again unwater the field, the neutralization and precipitation treatment of this water would assume huge proportions and entail tremendous expenditures.

However, our experiences in treating small volumes of acid water have shown that, when the neutralization costs can be borne and the necessary space for settling basins is available to accumulate the resultant hydrates, the remaining water from acid-mine drainage can be discharged into district streams without pollution.

The Davis Creek Dam

By M. N. Dunlap*

(New York Meeting, March 1947)

This article summarizes the successful incorporation of a flash-flooding stream into the tailing-disposal system at the St. Joseph Lead Company's Federal Division mill, in St. Francois County, Missouri.

The mill was built in 1906, by the Federal Lead Co., as its No. 3 mill for the concentration of southeastern Missouri lead ores, and damming Davis Creek was first considered in 1908. Plans were made also in 1925 and in 1937. Dam layouts varied from a reinforced concrete structure to a barrier built of tailings alone. All were abandoned because of liability that would be incurred by flash-flood rupturing of an incompleted dam.

By 1942 mill capacity had reached 12,000 tons daily and increasing tailing-disposal costs again focused attention on the Davis Creek basin.

This time the problem of riskless dam building was solved. Moreover, most of the items that appeared in the answer were materials that normally required wasting and the major portion of labor involved performed functions that it would normally have done, without constructive benefit, elsewhere. The new layout also made possible advantageous changes in mill-water supply pumping; static head being reduced from 514 ft to 95 ft, friction head from 4900 ft to 2860 ft, and pumping can be done now at periods of low power demand. The following facts about the Davis Creek watershed explain the 36-year reluctance to reach these, and other, benefits.

Size: 3600 acres.

Outline: roughly pear-shaped, 2½ by 3 miles at the largest dimensions.

Topography: mostly hill land with steep valleys.

Relief: 760 minimum to 1153 maximum.

Maximum rainfall: 3.36 in. per hour.

Permanent water: none natural. Now 2200 gpm discharged through 16-in. drill hole from mine pumps underground.

Geology: the 17 acres near dam centerline is Bonne Terre dolomite covered with sand and gravel eroded from Pleistocene beds that cap some of the hills. The remainder of the pond basin is on Davis shale.

Soil cover: mostly second-growth timber with some abandoned fields; all subject to careless burning, which reduces water-retaining ability.

Flood history: flash-flooding that had taken at least one life.

Downstream values: two railroad bridges before creek intersection with Flat River. Immediately below confluence, a highway bridge, a low-water crossing, a few business establishments and several blocks of residences.

In July 1942 the following segments of an ingenious plan were put into operation.

A ditch 270 ft long, 16 ft wide and 5½ ft deep was excavated to bedrock in the gravelly soil along the proposed dam's centerline. This excavation, which exposed many gravel-filled channels, was run full of slimes drawn from a riser set in the top of an adjacent tailings-disposal

Manuscript received at the office of the Institute May 9, 1946. Issued as TP 2176 in Mining Technology, March 1947.

* St. Joseph Lead Co., Bonne Terre, Missouri.

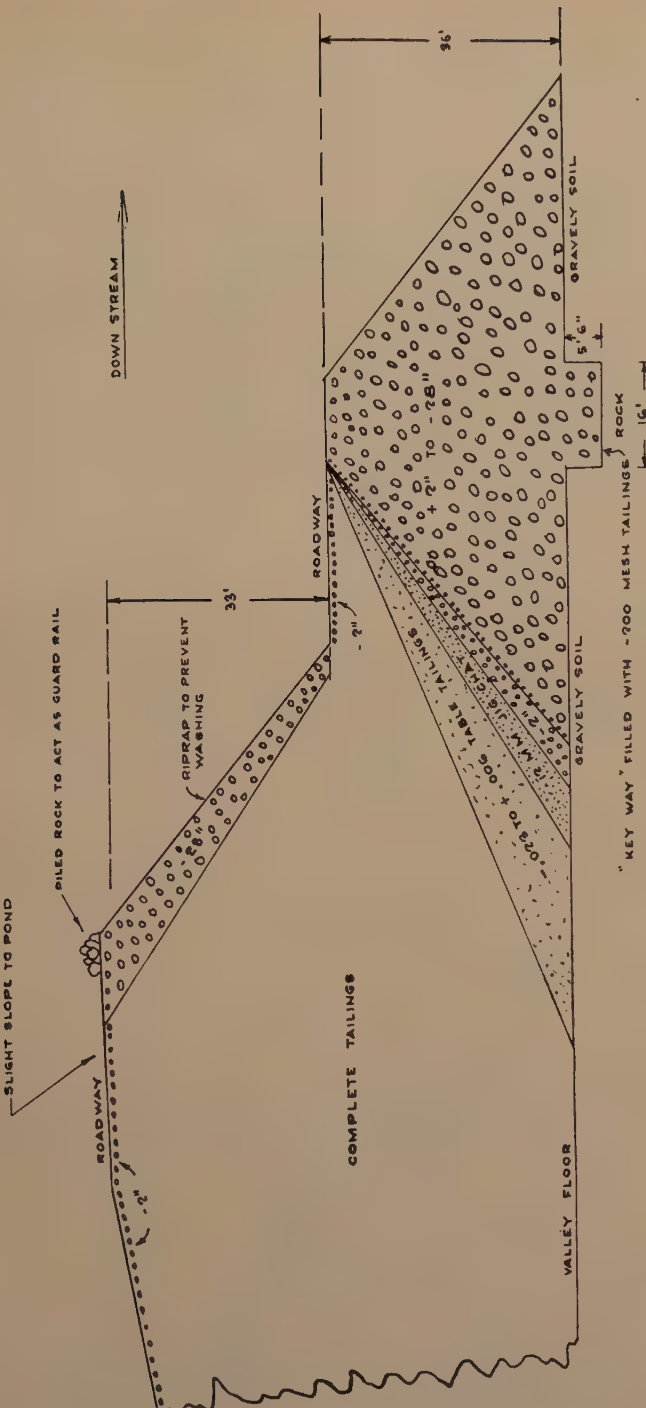

FIG 1—GENERALIZED CROSS SECTION, DAVIS CREEK DAM.

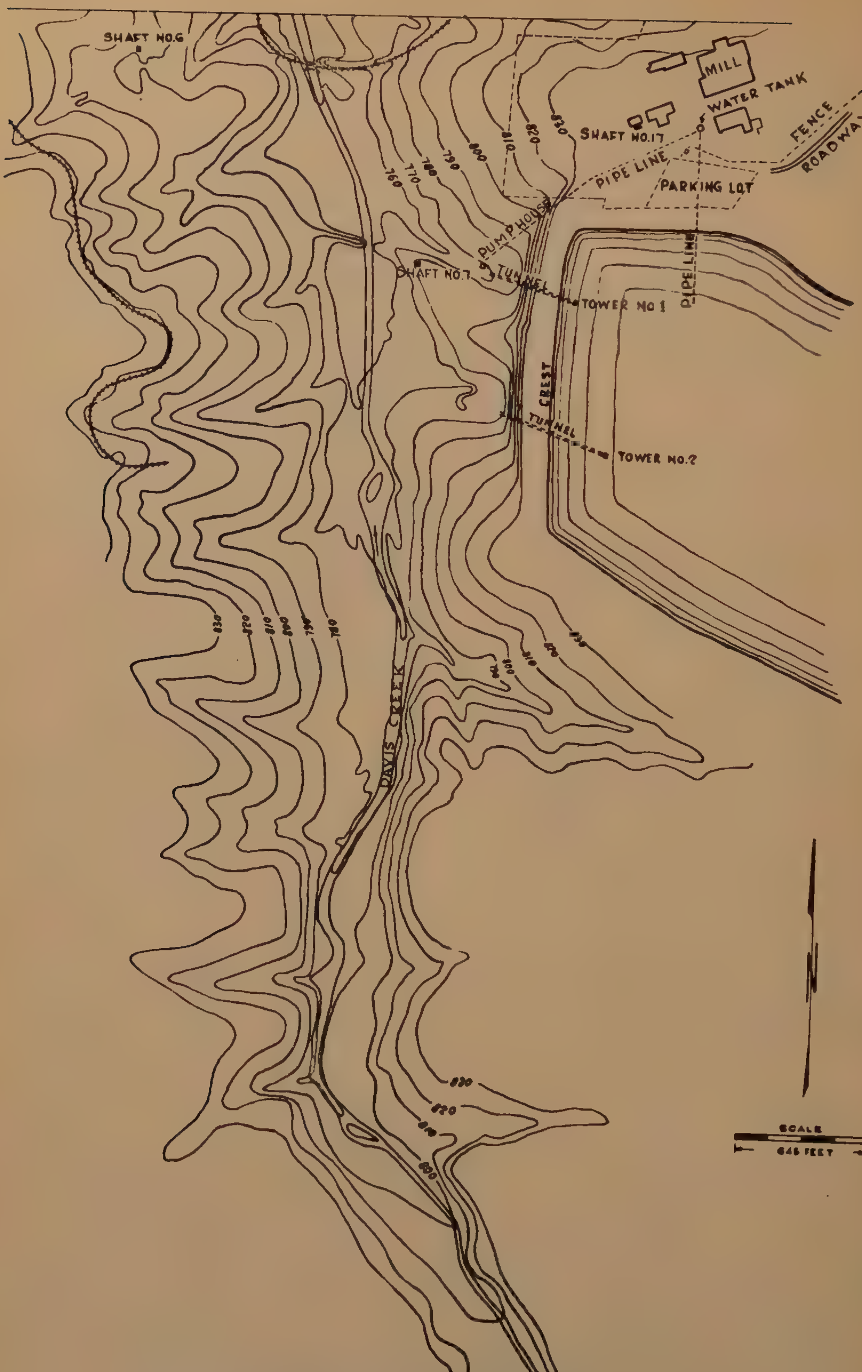

FIG 2—DAVIS CREEK SLIME POND BEFORE BUILDING OF DAM.

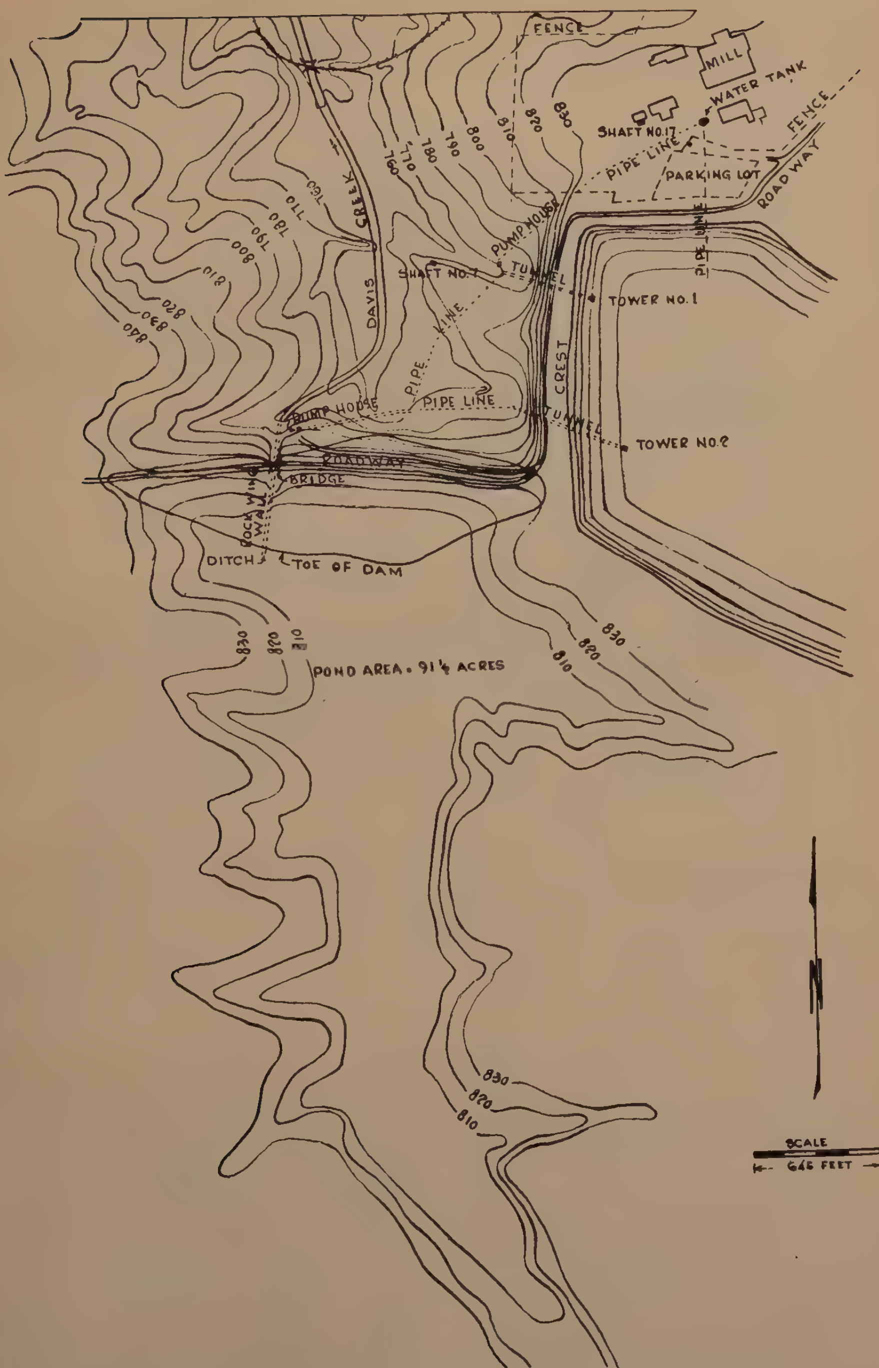

Fig 3—Davis Creek slime pond after building of dam.

line. Sod and debris were rooted from the residual clay that covered the remainder of the centerline.

A grizzly, with 2-in. opening, was placed in the bin at No. 12 waste-rock shaft and the plus 2-in. stone was hauled in dump trucks to the west end of the dam centerline, where it was used to make a level fill, at 797 elevation, 12 ft wide on top and 550 ft long. Maximum height was 36 ft and 24,600 cu yd of material was required. Round-trip hauling was 1¼ miles.

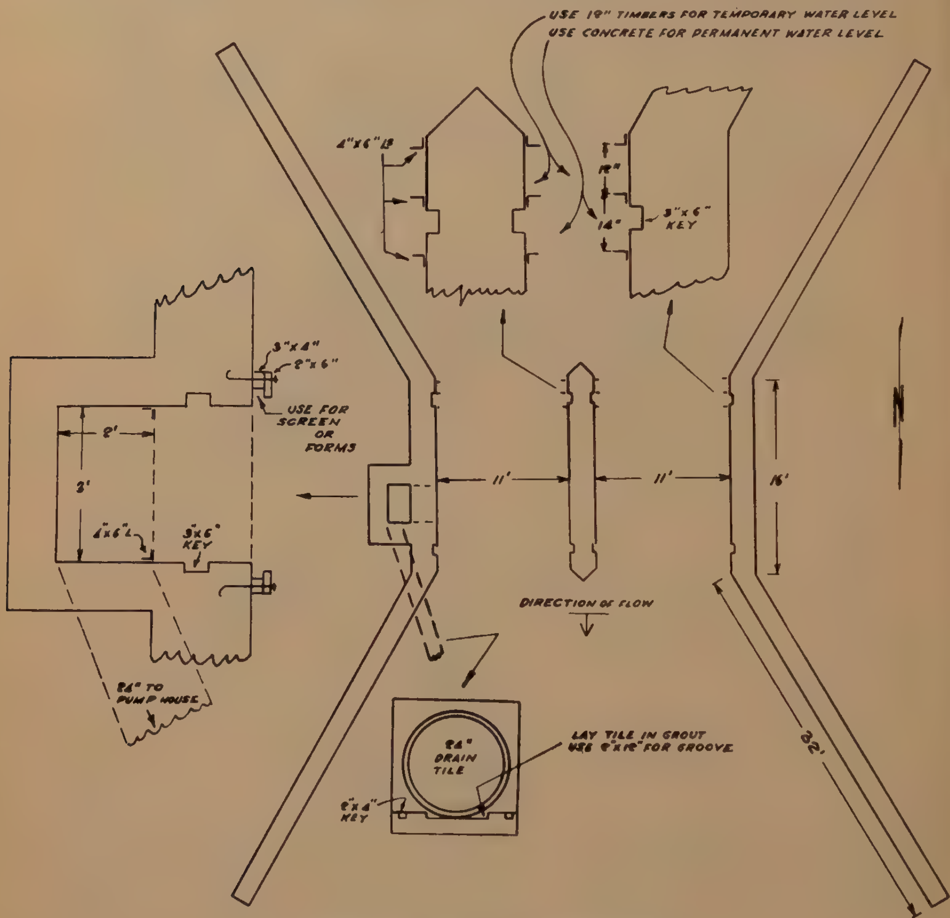

Fig 4—General plan, Davis Creek overflow.

It was the theory that the high percentage of voids in the screened stone would not only permit a rapid escape for any impounded flood waters, but that static head and velocity would drop so rapidly within the mass that little or no rolling or washing action would be present at the fill's downstream toe. A year's rains proved the soundness of this reasoning.

Sealing of the dam, which could be done whenever convenient, was delayed until the recovery-water tunnel and part of the overflow structure were concreted, but will be described now for continuity.

Minus 2-in. material was now dumped over the upstream side of the fill until all coarse rock was buried. This finer material

was in turn covered with jig chat from an old tailing pile. This sequence created a void size that increased downstream, thus ensuring a rapid decrease in static clevis made from ¾-in. rod. The bottoms of these tripod legs rest on pieces of 2 by 12-in. lumber and straddle the 10-in. pipe at approximately 40-ft intervals.

FIG 5—EQUIPMENT AND METHOD USED IN RAISING PIPE LINES FOR DAM BUILDING.

head for any leakage. A pipe line was then laid from the mill and, beginning at the east end of the dam, table tails were pumped (using clear water) to grade against the chat. The line was advanced westward until the entire south dam face was covered. From this stage dam raising was continued to 835 elevation east and 825 elevation west by the method developed by the Federal Mill organization. Approximate quantities of pumped materials are: 13,000 cu yd of table tails and 263,500 cu yd of complete tails.

Most of the equipment used for dam raising at the Federal Division is shown in Fig 5. The tripods are made by bolting three pieces of 2-in. pipe or discarded 10-ft diamond-drill A-rods together with a ¾ by 10-in. bolt and inserting a 10-in. The line is raised to full height by progressively lifting with 1½-ton chain hoists, where it is fastened to the clevis with an 8-ft piece of ⅜-in. chain equipped with a hook on one end.

Half-inch holes are then burned at 8-ft. intervals in the pipe bottom and covered with bands made from ⅛ by 3-in. mild-steel straps, to which are riveted linings ripped from wornout conveyor belting. The use of ½ by 6-in. bolts at the opening in these bands leaves a gap that permits the insertion of two opposing wedges made from 2 by 4-in. oak.

In operation the line is filled with water and feed is put on, then the tailings-disposal man, by loosening wedges, slips the bands from as many holes as can be operated without washing. Holes

Fig 6—General view of dam spillway and pump house.
Riprapped terraces can be seen on dam forming the pond used for storing mill water supply.

Fig 7—View taken from overflow structure.
Pumphouse in foreground, mill in the distance.

near tripods are not usually run until the pipe has been supported by filling at mid-positions. When tailings have filled to the pipe the bands are slipped over the holes and tightened by hammering the wedges.

If filling is desired on only one side of a line a dam is thrown up with a small gasoline crawler-type crane.

When the dam fill reached the required height, the top was shaped into a road bed by a bulldozer and surfaced with minus 2-in. material, both to provide traction and to prevent washing and wind erosion. The steep northern slope was faced with plus 2-in. rock to control washing and blowing.

For overflow and water-recovery purposes a ditch 800 ft long, 24 ft wide and, at one point, 30 ft deep was cut through the knoll at the west end of the dam. Some of the spoil was moved with a scraper hoist, borrowed from the mining department, some with a bulldozer, but the greater portion was loaded into trucks by Keystone Skimmer shovels and used for fill above the 800-ft elevation. Ditching, backsloping and the notching for the overflow structure amounted to 14,800 cu yd in the solid.

The overflow structure, which is really a combination bridge, mill-water-supply intake and pond-level-control device, is 45 ft high and required 400 cu yd of heavily reinforced concrete. The plan calls for lifting the bridge floor and filling the hollow central portion with waste rock, after a semipermanent water level is established and the side sections between piers are concreted to the proper height. The bottom of the recovery-water intake

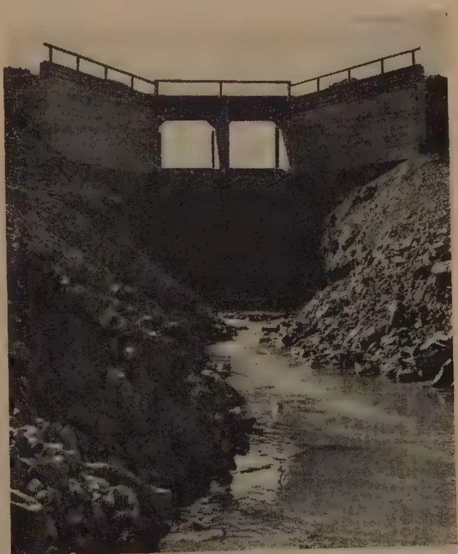

Fig 8—Reinforced concrete combination overflow structure, bridge and recovery-water intake.

FIG 9—RECOVERY-WATER PUMPHOUSE.
Transformers are 200-kva. The pumps are 3500 gpm driven by 200-hp motors.

is connected to the pumphouse by 255 ft of culvert made by pouring concrete around 24-in. drain tile.

The pumphouse is also built of reinforced concrete. It is 21 by 25 ft with an 8 by 14-ft bay on the south side for the electrical control panels. The three 200-kva. transformers, which step down the power for the two 200-hp, 3500-gpm pumps, are on the roof.

These two pumps are connected to separate 14-in. cast-iron pipe lines. One line discharges through a concrete bulkhead built in the former No. 2 overflow tunnel of the old slime pond, from which water flows by gravity into the mill; the other connects with an existing 14-in. line from No. 1 tunnel (1200 ft from the mill), and flow is directed by a butterfly valve, either to the mill or to the old slime pond. The No. 1 line is 1460 ft to the tunnel intersection; the No. 2 line is 100 ft longer.

Summarizing the advantages to the Federal mill:

1. The dam can be considered to be practically without cost because most materials used had to be wasted, and most of the labor involved would have been doing the same type of work elsewhere.

2. Ditching, concreting, extra hauling and equipment costs will be offset by power saved in mill-water-supply pumping, which is now being done at reduced heads during periods of lower power demand.

3. The 91½-acre tailing pond has an underwater volume of 73,545,500 cubic feet.

4. Maximum length of tailings-disposal lines is 4800 ft against a static head of 78 feet.

FIG 10—GENERAL VIEW FROM MILL-WATER-SUPPLY DAM.

Symposium on Grouting

(New York Meeting, February 1948)

CONTENTS

Introduction—Grouting in Mines

BY F. C. STURGES,* MEMBER AIME

BY definition the word "grout" means a thin mortar, or a kind of plaster or cement, and "grouting" means to fill up or finish with grout. The words "cement," "plaster" and "mortar" mean a substance that is used in a soft state which subsequently hardens. Thus, grouting is the filling of void space with a substance that hardens. Construction cements are the most common substances used, either by themselves or in mixtures, but the term grouting is applied quite correctly when other substances such as plastics and certain chemicals are used. Also, the use of the term grouting to cover the filling of voids with materials that do not actually become hard, such as the flotation tailings used in grouting in the southeast Missouri lead district, should be considered correct, as some stiffening does take place and, anyway, it is a better descriptive term than any substitute.

Engineers are interested in grouting because some of the materials with which they must work contain voids, and certain conditions can be improved if these voids can be filled with something solid and permanent. Grouting is done to accomplish several purposes: 1. To stop the passage of fluids and gases, 2. To cement materials together to make them stronger, and 3. To prevent consolidation of materials by filling void space with something solid as a replacement for the air and water that could be forced out under load.

The construction industry uses grouting extensively for all three purposes. Grouting is used to stop water percolation under and around dams. It is used to control water flow in tunnel driving and in shaft sinking. It is used to improve foundation conditions under civil engineering structures by permanently filling void space and thereby reducing settlement, or by cementing broken rock together and increasing its supporting strength.

The oil and gas industry does a lot of grouting, primarily to stop the passage of gases and fluids. The search for oil at increasingly greater depths has meant greater investment per well, and the oil industry has spent large sums on development of improved drilling methods. This

Manuscript received at the office of the Institute February 18, 1948. Issued as TP 2427 in COAL TECHNOLOGY, August 1948; MINING TECHNOLOGY, September 1948.

* Vice President, Pennsylvania Drilling Company, Pittsburgh, Pa.

research has resulted in the development of better methods and better materials for pressure cementing work on wells, and we are indebted to their engineers because these new developments are also available for other grouting work.

Mining men who extract their minerals in a solid state rather than a liquid state have some of the same problems to meet that their brothers in the construction and oil industries have, because they are working with the same basic material—the crust of the earth. This crust is full of cracks and crevices, and the mining engineer has to solve problems arising from the fact that these crevices usually carry water and weaken the roof and walls of his mine. However, the mining industry as a whole has not made as extensive use of grouting to solve these problems as do the construction and oil industries. This is probably because most mining men have had the opinion that grouting would be too expensive to consider. When it comes to discussing cost of grouting, little can be said except that the cost depends on the particular job. For instance, our charges for grouting have ranged from as high as $5.00 per cu ft injected to as low as 50¢ per cu ft, including furnishing the materials. Each job is a special case, and it should be pointed out that where a grouting job has a high cost per cubic foot the total cost of the job is generally low, and vice versa. The job mentioned above where the cost was so high per cubic foot was done with just a few sacks of neat cement, whereas the low cost per cubic foot on the other extreme was done with a mixture that was only 10 pct Portland cement and where the rate of injection was over 7000 cu ft per day. Where cementing action is important neat cement is generally used, but where the strength of the grout filling the voids is not so important, substantial savings can be obtained by using a mixture of a substance such as rock flour with a smaller amount of Portland cement.

The following table shows different substances that have been used for grout:

TABLE 1—*Grout Materials*

1. Cements
   1. *Portland Cements*—Regular Construction Cements—Grouting properties vary somewhat between manufacturers.
   2. *High Early Strength Cements*—"High-Alumina" Cements—Grouting properties include (1) Better penetration of small crevices because of finer grinding, and (2) Time saving because of early developed strength.
   3. *Quick-setting Cements*—Portland cements with an accelerator such as calcium chloride mixed in by the manufacturer.
   4. *Gypsum Cements*—Made from calcium sulphate. Set and harden to full strength at approximately the same time, in about 2 hr.
   5. *Special Cements*—Manufactured for special uses, such as:
      (1) Retarded setting cements developed for oil well use at great depth or at high temperature.
      (2) Iron oxide cements for high specific gravity and resistance to salt water.
2. Admixtures to Cement
   1. *To Accelerate Setting Time*—Calcium chloride (most commonly used), sodium silicate, sodium hydroxide.
   2. *To Retard Setting Time*—Sodium tannate, gypsum, lime, sugar.
   3. *To Bridge Crevices*—Bentonite, sawdust, grains, cotton-seed hulls, asbestos fiber, cellulose fiber, mica flakes, sand. A number of these substances are manufactured and sold under various trade names.
   4. *For High Angle of Repose*—Same as 3.
   5. *To Increase Plasticity of Grout*—Finely ground bentonite.
   6. *To Reduce Grout Shrinkage*—Finely ground bentonite.
   7. *To Lower Cost of Grout where Strength is Not Important*—Rock flour (usually obtained as a byproduct from a limestone aggregate plant), clay, ground shale, any fine material such as flotation tailings. Raw cement and sand have also been used but present difficulties in handling with ordinary piston displacement pumps.
3. Plastics
   1. Bakelite type plastics (phenol-formaldehyde compounds), injected in a molten state and harden on cooling.
   2. Asphalt.
   3. New synthetic resins or resin cements that harden from chemical action.

   Plastics have important grouting properties, such as resistance to contamination and ability to penetrate materials with low permeability.
4. Chemicals

   Sodium silicate mixed with calcium chloride, aluminum sulphate, or any heavy metal salt, plus, usually, the addition of an acid to control the time of setting. Useful for grouting materials with low permeability (as low as 0.0001 cm per sec.)

In some mines grouting has proved to be an effective tool in improving mine operation, particularly in lowering pumping costs, eliminating ice accumulation in air intakes, and improving safety conditions. In other mines it would not be worth considering. In order to aid mining engineers in understanding the latest develop-

ments and possibilities of grouting, this program was designed to accumulate information for study. We are greatly indebted to representatives of the construction industry, the oil industry, mine operators and grouting contractors for the information they are about to present to you.

Use and Technique of Pressure Grouting in the Construction Industry

By V. L. Minear*

This paper presents some of the problems encountered and solved by the construction industry during recent years while pressure grouting the foundation rock of dams. Pressure grouting has become "Standard Operational Procedure" and consequently many hundreds of thousands of bags of cement have been used for that purpose. There is nothing weird, mystical, or even particularly difficult about the basic idea. Holes are drilled in the rock and grout is pumped into them. This procedure is used to shut off the flow of water under the dam and to reduce the uplift on the dam. Like most other engineering and construction problems, the fundamental principle is simplicity itself, but some of the "minor details" are rather complex. When faced with the problem of actually forcing the grout into the rock, or keeping it there once it has been injected, an engineer frequently has need for all of the ingenuity and initiative that he has at his disposal. It is proposed to discuss these minor details at length, even at the risk of boring the reader. The difference between success and failure in a grouting program can well hinge upon some such detail that may not be fully understood.

Pressure grouting is a method used to make good foundations from bad ones by plugging actual or potential passages within the rock through which water might escape and thereby diminish or even destroy the usefulness of the dam. In some special cases, pressure grouting has been used for the purpose of consolidating broken rock to increase its bearing power. Lacking this useful method, many of the dams constructed during recent years could not have been built. Most of the really excellent dam sites in this country have been utilized long ago. Consequently it becomes necessary to find methods of utilizing the less desirable sites. Pressure grouting plays a vital role in the methods developed.

Engineers have learned the hard way, that the term "solid rock" is a misnomer. Solid rock does not exist to any considerable extent in nature. Practically all rock contains imperfections such as shrinkage cracks, bedding planes, joints, solution channels, folds, shear zones, crush zones, and the like. All these imperfections are weaknesses through which ground water is likely to flow. The function of rock grouting is to rectify these imperfections and make the rock as nearly "solid" as practicable.

The location, direction, depth and number of grout holes to be drilled are important. Clearly, a hole must pierce fractures in order to serve a useful purpose, for grout cannot be forced into the rock itself. Were it possible to determine the pattern and extent of the cracks within the rock, it might be feasible to drill two or three strategically placed holes so as to make practicable the grouting of the whole integrated network of cracks from them. Such is not the case. While a study of the geology, surface indications, and drill cores is helpful in gaining an idea of subsurface conditions, in the final analysis, the seams must be located by trial and error methods. There can be no doubt that the closer the

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spacing of holes, the more probable it is that all major seams will be intersected. Moreover, a close spacing of holes is conducive to economy of grout in that a shorter path of grout travel is afforded.

The ideal tool for drilling grout holes is the diamond drill. This tool produces a minimum of sludge and consequently a minimum tendency to choke the fine, hairlike cracks. However, this method of drilling is an expensive luxury when short holes are to be drilled under conditions where high grouting pressures can be used without danger of rock displacement. Percussion drilling is much cheaper and obstructions can be broken through by the injection of water under high pressure prior to the introduction of grout. This will be readily appreciated by anyone who has ever tried to calk a seam in rock from which water is leaking.

The equipment generally used in mixing and placing grout has been described elsewhere* but will be presented briefly for convenience. The grouting materials are mixed in either a specially designed grout mixer or an ordinary concrete mixer. In either case a suitable water meter should be used for measuring the mixing water since the measuring device with which most concrete mixers are equipped is not sufficiently flexible for use with grout.

Injection is by means of pneumatic displacement apparatus, by pump, or by gravity. Pumps are superior for placing neat cement grout but they are not suitable for use with grout containing sand or other coarse materials. Injectors handle such materials satisfactorily. Sometimes gravity alone is used in grouting shafts. Below is the weight of a column of neat cement grout, one inch square and one foot in height. From Table 1, the pressure that can be developed for any effective head may be computed.

* See *Civil Engineering* Nov. 1936, p. 751 and *Jnl.* of the American Concrete Inst., April 1947, p. 921.

TABLE 1—*Showing Unit Weight of Various Mixes*

*w/c**	Unit Wt in Lb	*w/c**	Unit Wt in Lb
0.75	0.798	2.00	0.614
1.00	0.736	2.25	0.598
1.25	0.692	2.50	0.584
1.50	0.660	2.75	0.572
1.75	0.634	3.00	0.562

* *w/c* measured by volume.

Actual grout injection is a highly skilled trade in which experience is of vital importance. Over thirty years ago, Mr. Robert Ridgeway, M. Am. Soc. C.E., stated,* "Grouting is expensive, and to secure the best results at the least cost and in the quickest time, it is wise to employ only those experienced in such work. To start with an untrained force usually results in discouragement and sometimes the abandonment of the work. After the men are well broken in, they seem to develop an instinct for doing the work in the right way, in the shortest time and accomplish what before seemed impossible." The author concurs entirely with this statement. The success of any rock grouting program hinges upon forcing cement into the offending voids and keeping it there. In accomplishing this work, the importance of experienced judgment cannot be overstressed.

Experience indicates that the amount of cement which can be forced into a given hole depends upon: 1. The character of the formation penetrated. 2. The consistency of the grout. 3. The pressure applied. 4. The rate of injection. These factors are closely related as will be shown later.

The character of the formation penetrated by a drilled hole, largely but not entirely, fixes the amount of cement that can be forced into it. Clearly, more grout can be forced into a rock in which numerous cracks, fissures, caverns, and the like, are present, than can be forced into a similar

* *Trans.* A.S.C.E. (1919–20) 106.

rock which is free from these defects; yet much can be done to increase the cement consumption of rock containing such defects to its benefit. Most of such defects, in their natural state, are partially, if not entirely filled with gouge or other material which must be removed before grout can be injected. Cement grout cannot be forced into mud for any appreciable distance. Moreover, any considerable mixing of the cement with the mud results in a structurally weak material lacking strength to resist the thrust or erosive power of the water in the seam. Incomplete removal of the mud before grouting often results in the water subsequently washing out the remaining mud, thus opening up a new channel, thereby obliterating any benefit derived from the grouting.

Drilling operations complicate an already complicated condition. This is due to a tendency for the cuttings to lodge in and obstruct the smaller fissures, regardless of whether percussion or rotary drilling is used. This fine, granular material acts as a filter which will allow water to pass, but will deny passage to cement or other suspended matter. This action builds a filter cake and reinforces the obstruction. However, the injection of water at high velocity, preliminary to the introduction of grout tends to open small continuous channels through this material by carrying the cuttings and gouge from the finer to the larger passages lying at a greater distance from the hole. Some engineers question the use of water for this purpose, particularly where a relatively tight rock is being grouted. Their objection is based on the belief that water may lodge in pockets and seams and occupy space which would otherwise be filled with grout. On the other hand, this objection may be answered on the theory that, under high pressure, the water will be forced into the rock pores or to other openings lying at a distance from the hole. If there is no such escape for the water, there is no need for grout in that particular crevice. Furthermore, there can be little doubt that when grout, under pressure, comes into contact with dry rock, this rock absorbs a considerable amount of the mixing water. This results in the grout becoming thickened, losing its fluidity and plugging the seam prematurely. For these reasons, the initial operation in grouting should be the introduction of water into the hole at the highest pressure which local conditions will permit. This operation should continue as long as there is any noticeable "pick-up," or increase in the pump's speed. Should the pumping operation cause an appreciable return of water from an adjacent hole, small "slugs" of compressed air should be introduced with the water in order to increase its turbulence and consequently its cutting power. This operation should continue as long as the return water is muddy.

The selection of a proper water-cement ratio is one of the most difficult and important phases of rock grouting. Adaptation of the consistency of the mix to the constantly changing conditions within the rock constitutes much of the secret of success in grouting. Undoubtedly more holes have been lost by inexperienced men attempting to use a thicker grout than the formation would tolerate, than by any other cause. Conversely, the use of a mix containing more than the optimum water content results in work of needlessly low quality. There is no safe rule for the determination of the mix to be used other than that it should contain the lowest w/c ratio that can be injected successfully. The pump's behavior, the rate of cement consumption and the reaction to various changes of mix are valuable aids in arriving at a correct solution. These signs are not infallible however and the solution depends largely upon the experienced judgment of the operator. The initial injection in each hole should be thin enough to enable the engineer to "feel out" the hole without the danger of losing it. This thin grout should be pumped only long enough to estimate

the hole's resistance and to enable the engineer to proceed intelligently with the adjustment of the *w/c* ratio. Since this adjustment consists of progressively reducing the water content to the minimum that the formation will tolerate, abnormally thin mixes do not give much information and tend to prolong the period of adjustment.

The ideal pressure to be used in placing the grout is the maximum pressure which can be applied without causing rock displacement. There is no inherent virtue in high pressure *per se;* the advantage lies in the fact that it makes possible the injection of higher quality grout through fewer drilled holes than is generally the case when low pressures are used. The flow of grout through the cracks and fissures within the rock is comparable to the flow of liquids through pipes. Frictional resistance is very high in the fine, tortuous, hair-like cracks and must be overcome by the application of pressure. It would seem that the higher the pumping pressure used, the greater would be the spread of the grout from a given hole and consequently the fewer the holes that would be required to grout a given area. This, however, is of secondary importance to the quality of the work done. It already has been stated that the thickest grout practicable should be used since the quality of the hardened grout is improved by the use of a low, original water content. The unit weight of thick grout may be almost twice that of a thin grout and consequently a correspondingly greater force is required to move it. When a needlessly low pressure is specified, a needlessly high *w/c* ratio is implied.

A close control of the rate of pumping is necessary. This is due to the tendency of the water to drop suspended cement particles unless a reasonably high velocity is maintained in the pump, delivery line, and rock fissures. When this occurs, not only is trouble experienced with the equipment, but the quality of the work is impaired by the loss of the coarser cement particles in the grout. These particles settle to the bottom of the horizontal reaches of the pipeline until the cross section is reduced to a point where the velocity is sufficient to retain all particles in suspension. A similar action probably takes place within the rock. Under these conditions, should there be an increase in the rate of pumping, these coarse granules would probably be picked up, thicken the grout and possibly plug the hole.

The foregoing discussion is largely academic and is intended to explain some of the relationships noticeable during grout injection. The discussion reveals three inter-related variables: applied pressure, water-cement ratio, and pump speed. A change in any one of these variables will be reflected in one or both of the others. Thus, decreasing the water-cement ratio is accompanied by an increase in the indicated pressure and a reduction in the pump speed. Similarly, a higher speed with constant *w/c* results in a higher gauge pressure. It is necessary therefore, to coordinate the three variables. This is accomplished in practice, by maintaining the pressure and speed nearly constant, by manipulating the *w/c* and, to a lesser degree, the throttle. An effort should be made to pump at the maximum allowable pressure at all times. In case it becomes necessary to race the pump in order to obtain this pressure, the grout should be thickened until a reasonable speed is reached. If the pump labors at the required pressure, the grout should be made thinner.

Another "minor detail" which frequently causes major difficulties, is retention of the grout within the rock into which it is being injected. When pressure grouting operations are properly conducted, the liquid grout finds and escapes from any existing opening in the same manner that air escapes from an automobile tire. Patching operations are much more difficult with grout leaks, and some form of surface

treatment is usually necessary to prevent the escape of grout when pressure is applied. This treatment may range from simple calking operations, through so-called "stage-grouting" to concrete lining.

The ability to seal leaks quickly is of prime importance. Costs, due to excessive leakage, can be very heavy. Unless the grouting operator has sufficient information to enable him to effectively stop the flow, the grout slips away from him and he is powerless to prevent it. He finds himself confronted with a situation which requires the grout hole, costing perhaps several hundred dollars, to be abandoned or the pumping operation continued with much of the cement leaking to waste. Discontinuing pumping, even temporarily, is dangerous. This is particularly true during hot weather and it frequently results in the loss of the hole. In any event, the quality of the grout job is damaged due to the segregation of the cement particles which begins immediately after the flow is stopped. Abandonment of the hole often results in a recurrence of the same leak when adjacent holes are grouted, so that the net result may be the abandonment of several holes due to a single leak and an admittedly unfavorable condition ignored.

Since grout leaks can be expected at any time, it is almost mandatory to use adequate equipment, a trained crew and experienced supervision. When it becomes evident that a given method will not stop the leak encountered, sufficient materials should be available to permit various methods of calking in rapid succession.

The calking materials ordinarily used are oakum, lead wool, wooden wedges and flash-set cement. The cement is considered the most useful and versatile.

The best flash-set cement known to the author consists of a blend of 1 part Lumnite cement to 2½ parts of Portland cement. Properly combined and used, this blend has the faculty of taking its initial set in 5 min., its final set in 20 min. and has a compressive strength of 700 psi in 30 min. It can be used in either its dry state or its liquid state. In either case, exact proportioning and an intimate blend is essential. If desired to use the dry blend for such purposes as applying patches to leaking surfaces, or for guniting wet areas, the proper amounts of the dry cements are mixed in a concrete mixer after which they are resacked for future use. If it is desired to use the blend as a grout for cutting off flowing water, grouting in pipe connections, or for stemming for blast holes, the flash-set grout should be made by combining Lumnite grout with Portland cement grout, stirring vigorously for not more than one minute and depositing the blend immediately. Not more than equal parts of water and cement by volume should be used and the Lumnite grout should be poured into the Portland grout.

An excellent description of the use of this material is contained in an unpublished report, dated Oct. 1, 1936, by F. A. Backman, on the subject of "Field Methods for Pressure Grouting of San Jacinto Tunnel (Metropolitan Water District of Los Angeles)" which is given in part below:

"The grouting of the battery station at Cabazon resulted in the cutting off of approximately 650 gpm between this point and a little east of there as measured by the hydrographer. The battery station is a drift running into the left wall about 400 ft east of the beginning of the Pioneer drift. The purpose is to maintain the locomotive repair, battery charging, and such, near the heading. Considerable water and bad ground requiring timber were found on advancing this drift. Its use as a repair shop necessitated grouting and guniting so the two operations were more or less combined. Consideration is here given to the methods used. Long holes were drilled to intercept the water courses as far from their opening into the drift as possible. As many holes were drilled as were necessary

to take the water away from its original outlet. Pipes were placed in these holes and calked in with lead wool. The open seams in the rock were then patched with a quick setting mixture of Lumnite and Portland cement. This sealing of the open seams was not entirely successful because of the inaccessibility due to the timber and lagging, but was an aid in retaining the grout in the rock. Additional holes were drilled into the seams near their original outlet, and into these holes a thick mixture of sawdust, aqua-gel and cement was pumped until it began to flow out of the pipes then discharging the water. This effectively sealed all but the finest crevices and permitted pumping at high pressure when the final grouting was done. Before attempting to grout the pipes containing the heavy flows of water it was deemed advisable to gunite the more shattered areas to give additional strength to the ground before applying the grouting pressure necessary to secure a satisfactory job. A certain amount of preparation was necessary before gunite could be placed as there were numerous small leaks that had to be piped before gunite would adhere to the rock. Short holes were drilled and ¾-in. nipples inserted to relieve the pressure while guniting. One part Lumnite and three parts Portland cement were mixed with dry sand (ratio 1 cement to 3 sand) and shot in the usual manner. The three to one Portland-Lumnite mix hardens in less than 5 min. and permits guniting in comparatively wet areas that with ordinary cement would be impossible. After guniting, the ¾-in. pipes were grouted with cement grout placed by air through a ¾-in. hose using approximately 100 lb pressure. The larger flows were grouted with the use of Gardner-Denver pumps and were pumped to refusal at 1000 lb pressure. Most of the lagging was then removed from the timber sections and the entire area gunited to a thickness of from 1 to 5 in.

Mixes for Use in Cement Gun Only

Lumnite	Portland	Lime	Sand	Temp Water °F	Set	High Strength
½ . . .	½	0	3	90	1 min	2 hr
½ . . .	½	0	3	65	10 min	5 hr
1 cf. .	0	5 lb	3	90	1 min	20 min
1 cf. .	0	5 lb	3	65	10 min	2 hr

These are approximate mixes and will vary slightly with temperature conditions especially that of the aggregates."

Successful grouting consists of devising a technique which will satisfy requirements of the local geology, conditions, and the like. There is given below a bibliography listing articles in which are described various problems solved by the construction industry with pressure grouting methods. The interested reader is referred to these articles for further data on the subject.

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Technique of Pressure Cementing in the Petroleum, Mining, and Construction Industries

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Part I

In the petroleum industry, the process known as oil well cementing is the equivalent of pressure grouting in the mining and construction industries. The science of oil well cementing has been known and practiced for many years. At the present time it has been brought to a high degree of perfection in the handling of many different types of jobs, all of which are known as oil well cementing. In order to understand more easily these processes, we will discuss briefly several different types of oil well cementing jobs, the purposes for which they are performed, and the methods of their accomplishment. It is believed that some of these procedures, in a modified form, can be applied to pressure grouting projects in general.

Originally, oil well cementing consisted of placing a neat cement slurry about a steel casing set in a bore hole. Cement is used about these casings to protect them from the corrosive effects of ground waters, to protect the oil and gas zones from encroachment of undesired water from other levels, to prevent the blowing out of high pressure oil and gas zones from shallow levels, and to strengthen generally and support the casing being placed in the bore hole.

After the bore hole is completed to the desired depth, casing is run into the well, pumps are attached to this casing and circulation of the mud fluid in the hole is established; this circulation being from the inside of the casing into the annulus about the casing, and thence back to the surface. Next, a cement slurry is mixed and pumped into the inside of the casing string. A plug, which acts as a piston within the casing, is placed above the cement slurry. This is to separate the top of the cement slurry from mud or water which is used to shove the cement down through the casing and out into the annulus. This plug further serves the purpose of shutting off against a baffle in the bottom end of the casing string, thus stopping the flow of cement out of the casing and leaving the shoe, or lower end of the casing, surrounded by cement and also filling the annulus there above. Casing strings cemented in the above described manner may vary in length from a few feet of large diameter pipe to very long lengths of smaller pipe. As of the present date the longest string of casing which has yet been cemented is some 16,600 ft deep. As a general rule, the deeper the well and the more cement slurry used, the higher the pressures which are required to place the slurry. In parts of West Texas, it is not uncommon to use 4000 sacks to cement a single string of casing in place.

Machinery used to mix the cement and place it into the casing is mounted on heavy duty, powerfully equipped trucks which are capable of negotiating any type of highway or country roads. These trucks

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are equipped with two or more pumps which may be either power driven from the truck engine itself and auxiliary engines, or, where steam is available, the pumps may be of the steam type and use steam from the rig boilers. There are many types of these cementing units, ranging from very small sizes for use in areas where small amounts of cement and low pressures are encountered, to very large units equipped with twin diesel engines, and capable of producing pressures of more than 10,000 psi.

The slurry is usually mixed by a hydraulic jet mixer which is composed of a conical funnel ending in a bowl shaped like a pipe elbow. In the back side of this bowl is located a small jet through which water is pumped at fairly high pressure. This jet action produces a vacuum in the funnel drawing the cement down into the stream of water and mixing it very thoroughly and rapidly by agitation in a short discharge pipe from the elbow. This cement slurry is then collected in a small vat where it is picked up by the suction of the pump which places it into the well.

Primary cementing jobs, as described above, have grown into many different forms, some of which may be applicable to grouting in a manner that the primary job is not. The principle type among these is known as "squeeze cementing." By this procedure, cement slurry is pumped down into a well through a small string of tubing or drill pipe and forced out into various zones in the well which are carrying undesirable water or gas. It has been established that cement, under high pressures, will penetrate into sand zones, probably by producing crevices therein, along the bedding planes, and in so doing will shut off undesirable water or gas. Many jobs of this type are done every day and are attended by a high degree of success in cutting off these undesired fluids.

In squeeze cementing there must be some pressure applied to the well bore to cause flow outwardly into the formation to be cemented. Filtration of the slurry by the permeable formation with which it is in contact causes the water in the slurry to pass out into the permeable medium leaving in place in the hole and in crevices, a very dense mass of cement in the form of a filter cake against the face of the permeable zone. It has further been found that if sufficient pressure is applied, the dehydrated slurry will obtain very quickly a high degree of strength, and will remain in place and not move on the release of the pressure applied to the zone.

These differential pressures in wells frequently run very high, as much as 9000 psi pump pressure has been applied; however, in shallow holes, and in highly porous and permeable zones, quite low pressures may be sufficient to allow for injection of large amounts of slurry. In any case some dehydration of the slurry toward the conclusion of the injection process is desirable and to accomplish this, a pressure higher than that required at the beginning of injection should be obtained.

The equipment necessary for oil well cementing is relatively simple to those who are experienced with it. It has been brought to a high degree of perfection and can be built along varied lines to fit any operation which might be necessary. It is entirely possible to produce equipment such as that mounted on the heavy trucks used in the oil field in smaller, more compact forms which could be lowered into a mine shaft, or to be used in tunnels. These pumping units could be equipped with various types of power, for example, electric motors instead of the diesel engines commonly used in the oil industry.

The hydraulic jet mixer is the least complex, speediest mixer available for the handling of large volumes of cement as are required in oil well cementing. This mixer is capable in one of its forms of mixing approximately 50 sacks of cement per min, and in smaller forms can be modified to

mix a maximum of 15 to 20 sacks per min. This mixer can further be modified to produce cement slurries with varying water-cement ratios, particularly those of very high water-cement ratio, as are needed in most grouting procedures.

Numerous materials are used in oil well cementing, by far the most common of which is ordinary Portland cement mixed only with water to form neat cement slurries of varying water-cement ratios. For many years no other form of cement was available but as specific problems arose other cements and materials were developed to meet the required conditions.

High early-strength cements were introduced to save time in waiting for the cement to set up in a satisfactory strength. These cements are ground more finely than regular Portland cement and in some cases also have chemical differences. Ideally, they are characterized by pumpability periods about equal to those for Portland in the lower temperature ranges but acquire their final setting periods much more quickly than Portland and attain a strength in 24 hr about equivalent to Portland over a 48–72 hr period. These cements are essentially time savers which is a matter of considerable economic consequence.

As well depths become greater the time for cement placement and the temperature to which the cement is exposed increase rapidly. This caused introduction of the retarded or slow setting cements. A cement of this type is nearly always ground somewhat coarser than Portland and has mixed into it small percentages of various groups of chemicals which assist in slowing down the initial setting time. While the initial setting time is retarded and the pumpability period greatly increased at any given temperature, the final setting time for such cements is not materially changed from that of regular Portland and further the 72-hr strength of both types is about the same. Use of retarded-set cements is indicated in most instances where the temperature to which they are exposed exceeds 180°F.

Where highly mineralized ground waters are encountered, which are high in sulphates, most cements offer only limited protection as sulphates affect them rapidly. To combat these waters, special sulphate resistant cements have been developed and in some areas are widely used.

In most cases, regardless of the type of cement used, the mixed slurry is only cement and water, yet frequently various admixes are used in the slurry for several reasons. Very finely ground bentonite in small percentage is added to increase slurry viscosity, to prevent settling and to allow the production of light weight slurry with high water-cement ratio. Chopped cellophane flakes and certain other materials may be added to a slurry to cause bridging over of crevices and large porous areas which would otherwise rob the slurry away from the bore hole or cause loss of returns during the cementing process. To produce an inexpensive, very rapid hardening cement, calcium chloride may be added to the slurry; however in the use of this material caution is required, especially if temperature is in excess of normal atmospheric conditions. In cases where radioactive surveys are to be made to determine the final position of the cements, small amounts of carnotite may be added to the slurry.

Recently, there has been considerable emphasis on still further improved cements for oil well cementing. Much work has been done on a material known as resin cement, which basically is ordinary cement mixed with a water soluble synthetic resin. This particular material has not yet been completely developed, but has already shown a high degree of promise for squeeze cement work. When this mixture contacts a permeable zone, the liquid phase is squeezed out into the sand body in such manner that the liquid plastic will harden in place and seal off between sand

grains. It can be proved in the laboratory that it is often impossible to force neat cement slurry into the interstices of a sandstone, because the cement particles are as large or nearly as large as the interstices themselves. However, if the pores of the sand are of sufficient size, it is frequently possible, with a high water-cement ratio to wash out into the sand enough cement to produce a sealing action. In the case of resin-cement slurries, the liquid resin and water phase easily penetrates the sand body without producing an early filter cake of cement and will harden in place while the remaining solids resembling the regular cement will harden in place in the larger openings and against the faces of the permeable zones. This material is of a critical nature with regard to temperature and must be balanced before use for the temperature of the zone in which it is to be placed. At the present time, these materials are quite expensive, costing approximately $20.00 per cu ft of slurry used. It is believed, however, that the cost of these special products can be reduced as supplies of the proper material become more readily available.

In some instances a synthetic liquid resin may be used without being mixed with the cement or any other filler. This is particularly applicable in dense sandstone where it is difficult to seal off the undesired water or gas by the mere cementing of the surface of the zone. In this case, the true fluid is easily placed back into the interstices of the sand. A liquid resin without admixes may be pumped great distances into the sand zone, and will disseminate through the sand in all directions. Therefore, sometimes control is necessary to prevent the material getting too far away from the bore hole or to prevent it seeking the more permeable zones rather than to go into all zones equally. A method has been worked out and an admix produced for liquid resins, so that they now may be placed across zones of varying permeability and yet insure an adequate flow into all sections of the zone regardless of its permeability. It is possible that the use of both the straight resins and the resin cement combination may have considerable application to grouting problems.

Another material which is frequently used in oil wells, is called Hydromite. This is a gypsum base material, similar to plaster of paris, to which has been added a dry powdered synthetic resin. Since Hydromite is capable of setting under continued agitation, and also can be controlled to solidify at various setting rates, such material is advantageous in the cutting of a fast flow of water.

Similar to Hydromite but not containing the resin admixture is Calseal, a highly refined gypsum cement. It is used in much the same manner as Hydromite but must be regarded as a temporary material since it is rather rapidly deteriorated by various ground waters. Calseal is widely used for the tamping of nitroglycerin and other explosives in oil well shooting.

Part 2

As mentioned earlier in this paper, many oil well cementing techniques are adaptable to pressure grouting. Likewise much of the equipment can be used for this work either as now built or with some modification. Regular materials used in oil well cementing are also the same as those used in pressure grouting and it is probable that new and specialized materials being developed may find a place in the latter type of service.

A large percentage of pressure grouting projects are remedial in nature. Some typical examples of this type of work are:

Dam Repair.

Building foundation stabilization.

Mud jacking concrete slabs and roads.

Mine water control.

Frequently a thorough study is not made of the foundation upon which a dam is to be built. This is particularly true of

smaller municipal and privately owned projects. As a result, serious leakage may occur either under the dam, around the ends of the dam, or through the structure itself. Pressure grouting has been very successful in correcting these conditions.

If the fissures are small, a high water-cement ratio, neat cement slurry is injected. As mentioned in Part I of this paper, only a slurry of this type will penetrate such cracks and crevices probably because of a wide dispersion of the cement particles themselves in the slurry. Such dispersion reduces the bridging effect of the grains at the crevice opening, allowing them to enter in sufficient quantity to finally form a dehydrated mass that will withstand the injection pressure and give the desired sealing effect.

As the size of the opening increases, the water-cement ratio is decreased and admixes are added. Bentonite may be added to increase the viscosity of the slurry. Should a rapid set be desired, calcium chloride may be added.

If the cavities are large enough to require bridging before the slurry will remain in place, such fibrous material as cottonseed hulls or shredded cellophane may be added. In extreme cases, it may be necessary to gravel pack the cavities.

The use of various admixes must be tempered with good judgment. No specific rule can be set up as to type or quantity that is required since local conditions are the governing factors. However, it is important that such admixes do not retain the slurry to the area immediately adjacent to the injection hole only. This point is the major difference between oil well squeezing and most pressure grouting projects. In the former case, slurries are usually injected to protect the well, while in the latter case, the hole acts only as a point of entry to the formation.

A special gravel packing technique was developed on a dam repair job at Council Grove, Kansas, in June 1947. On this job, water was flowing under the dam through a limestone formation in which was found numerous mud seams and large cavities. There was a head of 50 ft on this formation with a very high water velocity. Heavy slurry with fibrous admixes was only partially successful and an excessive grout loss was experienced. One key hole was chosen with a 3-ft cavity. Very coarse sand and fine gravel were washed into the hole through the grout pipe, first with water and later with cement slurry. A total of 89 yd of this material was lubricated into the hole, water flow was substantially reduced and eventual refusal at 60 psi pump pressure was obtained.

Under certain conditions, building foundations may be stabilized by the injection of cement slurry. This process is especially adaptable to structures built on unconsolidated sands and gravels fully saturated with ground water. The technique may require the setting of grout pipes through the entire unconsolidated section, gun perforated at the proper levels and the slurry squeezed out into the sand, or it may involve the driving of the pipes only partially through the section, open ended, and forcing the grout out the open end upward and around the pipe allowing it to seek the more permeable zones. A combination of the two methods may also be used. In either case, the problem is to force cement outward from the point of injection through the more highly permeable zones in horizontal layers segregating the finer and less permeable sections, thus reducing either vibration or settlement.

If vibration set up within the structure extends downward into the sand, there always appears to be a reclassification of the material into fine and coarse layers. This reclassification becomes progressively less, the deeper the sands are penetrated and usually becomes negligible at a depth of 3 to 4 ft. Thus, the forming of horizontal layers of set cement, a short distance below the mat upon which moving machinery has

been mounted, will stop the vertical movement of the sand grains and tend to reduce both vibration and settlement.

The greater the mass of cement injected, the greater is the assurance of success of the project. Therefore, some experimenting must be done with water-cement ratios to determine the proper proportions. Except in cases of very coarse sand and gravel, water-cement ratios of 20 gal per sack or more are normal. One case on record, namely The Kansas-Nebraska Gas Co. station at Deerfield, Kansas, required slurries with a minimum ratio of 30 gal per sack and at times it was necessary to increase this ratio to as high as 75–80 gal per sack. Lower ratios would immediately seal the exposed face of the sand preventing the injection of any slurry.

Since the slurry injected into unconsolidated material normally assumes a lenticular form in the more permeable zones, this type of work does not lend itself to the building of a grout curtain around a predetermined area unless the material is quite coarse and uniform throughout the section. It is probable, however, that a curtain could be accomplished by the injection of a material other than cement slurry. The use of synthetic resins has been considered but as mentioned earlier in this paper, the cost of the material usually is prohibitive.

The technique of gun-perforating grout pipes was especially developed for a foundation job performed for the Cities Service Gas Co. in 1941 at their Wichita, Kansas, pump station. In using this technique, a small gun, similar to the one used in deep wells, is lowered into the grout hole and fired at predetermined levels, perforating the pipe with either $\frac{3}{8}$- or $\frac{1}{2}$-in. steel bullets. Since that time, it has been successfully used on numerous projects. The number of perforations and their spacing depend upon local conditions.

Floors or slabs, poured directly on the ground, frequently sink below grade due to settlement of the underlying material. This condition may be corrected by forcing a slurry between the slab and the surface upon which it lies. This type of work requires very low injection pressures and care must be exercised so that the slab is not lifted above the desired grade line. Whenever possible it has been found advantageous to carry out the lifting operations in stages, thus reducing the stress on the slab. It is also important that the voids created be entirely filled with the injected slurry.

In the field of mining engineering, the problems of shaft sinking are comparable to those encountered by the petroleum engineer in the drilling of deep wells. One of the major problems in both instances is water control. This problem may be encountered either while the work is in progress or after completion.

Water intrusions after completion have been reduced or completely stopped by injecting cement grout through holes in the shaft walls or through grout holes drilled around the outside of the shaft. The first method is usually more effective if the walls have sufficient strength to withstand the necessary injection pressure. However, the physical difficulties encountered in making the set up often make it undesirable.

In some areas it is known from previous experiences that serious water difficulties will be encountered while sinking a shaft. Pre-grouting the shaft area will usually control this water sufficiently to allow the work to progress without expensive delays. Normally, grout pipes are set above the first water zone and a stage grouting program is carried out until each zone has been properly sealed. It also seems probable that the grout pipes could be set through all the waterbearing zones and selectively gun-perforated for squeezing. However, to our knowledge, this procedure has never been used.

The above methods of pre-grouting mine shafts are effective in hard rock areas only.

Attempts to pre-grout quicksand before shaft sinking have proved unsuccessful. It is believed that the use of straight resins or resin-cements might solve the problem in such areas if the additional costs of material could be justified.

Pressure grouting has also been called upon to do various miscellaneous remedial jobs ranging all the way from plugging old conduits to filling rat holes under grain elevators.

It appears that perhaps the greatest advantage the mining and construction industries can obtain from the oil industry in their grouting problems, is in the use of the special materials described, together with the multiplicity of tools in the way of packers, gun perforators, hole calipers, cement mixers and pumps which have been developed for use in oil well cementing and other phases of the petroleum industry. It seems likely that these devices can be applied to certain phases of grouting, especially so in both pre-grouting and remedial work on mine shafts and dams.

An Example of Controlled Pre-grouting in Shaft Sinking

By R. H. Allen* and J. W. Galpin,† Members AIME

Controlled pre-grouting is a technique developed during a period of more than ten years experience, in an effort to produce safer, drier and more economical mine shaft sinking. The technique involves a knowledge of hydro-mechanics, both static and dynamic, and a series of specially formulated grout mixtures to obtain several desired results through precontrolled setting time.

In general, the following procedures have been used to secure the best results in controlled pre-grouting:

1. Before drilling is commenced, one of two decisions is made: one procedure, a modified form of the Francois method, is to stage drill and grout, that is, drill a limited portion of the hole, then grout and redrill that section which has been grouted, progressing in stages until the required depth is reached. We prefer this method because no casing or grouting packers are required in the bore hole. This eliminates the risk of losing them in the bore hole due to packer failure, which might result in the expense of drilling a new grout hole. Another feature of this approach is that smaller drill holes may be effectively used, further reducing the costs. One drawback, however, is that because of the open and small size hole, it is impractical to attempt water draw-down tests.

The second alternative is to drill the bore hole, (generally NX size), to the required depth. The casing and grouting packer are placed in the bore hole at the lowest packer setting. A small, portable, power driven, swabbing device is used in the grouting casing to test the packer and remove mud or cuttings from the bore hole below the packer. The hydrostatic level is recorded and a close estimate or test is made of the quantity of water at each packer setting. Grouting progresses upward in a series of packer settings to the surface. Occasionally, casing is inserted in the bottom of the bore hole from within the mine, and water pressure and volume can be obtained in this manner. This is usually dangerous because of the unknown quantity of water.

2. The grouting, mining and geological engineers at the drill location make a detailed examination of the cores and also locate proper and firm grouting packer locations. A competent driller's opinion should be given due consideration as to the character and condition of the bore hole.

3. Several samples of formation water

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are taken from the bore hole at various depths. A pH test is made of each sample, and if necessary, a chemical analysis. Should any of these indicate sufficient chemicals to damage cement or its setting time, then preparation is made for storage and transportation of neutral water, or in some instances, special cements are ordered.

4. While the above hydraulic or drawdown tests are being made, the grouting engineer gathers several dry samples of the various cements and other grouting materials on hand which he expects to use within twenty-four hours. The samples are prepared with the mixing water to be used to obtain several different controlled setting times. Emphasis should be placed upon reproducing the approximate formation temperature and hydraulic pressure range of the sections to be control grouted. It is generally true that cold water retards and warm water accelerates the setting time of Portland, gypsum and aluminous cementing materials. Particularly is this important in controlled grouting. During the processing of these tests, others are made as follows: weight per gallon of grout mix (water-cement ratio), color, as some samples are made with various tracer dyes, viscosity, and the specific gravity. The tests are carefully recorded, and each is important in the technique of controlled grouting.

5. The last step before proceeding to grout is to pump water into the grouting packer point below the packer to record the hydraulic pump pressure at various injection rates in gallons per minute. A high pressure water meter is used in this instance.

6. I have selected, for this paper, an example where the use of conventional grouting methods indicated failure and was followed by the successful application of controlled grouting.

Location: West Virginia.

Proposed shaft: 18 × 18 × 500 ft vertical.

Proposed drilling pattern: NX size—9 holes—1 in the center and 8 in two concentric circles around the shaft site. No. 1 SE bore hole was grouted first on the proposed shaft line. The general character of the formations are gravel, sandstone, broken fissured shales and slates, interbedded with sand streaks. The formations usually contain subsurface water flows with interformational movement of fluids.

The depth of No. 1 SE bore hole is 517 ft drilled without any interruption for grouting. The casing and packer were run and expanded to the walls at 435 ft. Water was pumped into the formation to determine the operating pressure and found to take water at 22 gpm with 345 psi. This was followed by 155 bags of cement of unknown weights and viscosities, and with the use of no additives to increase the viscosity nor accelerators to hasten or control the set. The same procedure was used in the entire grout hole at five different packer settings. 1472 sacks of cement were pumped into the No. 1 SE bore hole with a maximum static pump pressure of 385 psi.

The drill machine was moved 27 ft to No. 2 NW bore hole and drilled to 485 ft. No cement grout was found in any of the cores recovered from the grout hole. The bore hole was grouted in the same manner, and with the same corresponding packer settings as No. 1 SE. The pump pressure variations, at the same grout injection rates, were unimportant. 1187 sacks of cement were used in this grout hole. The total number of sacks of cement in the two bore holes was 2659. It was then decided to redrill No. 2 NW and test for results. The test was made from within the mine, and indicated the bore hole to be making more than 250 gpm of water. The bottom of the bore hole was then plugged off and grout control started with the same five packer points. Five specially formulated batches of controlled flash setting, expansible grout were prepared with various tracer dyes—each batch being of 50 sacks,

a total of 250. In conjunction with this, during the five grouting stages, 600 bags of cement grout with controlled viscosities and weights were also used. To fit individual grouting conditions, various angles of repose from 12 to 42° were necessary. The normal angle of repose of mixed neat Portland cement is 5° with no additives. Bentonitic colloidal clays were used to increase the viscosity with a resultant higher angle of repose. Bore hole No. 2 NW was regrouted completely with controlled grouting from bottom to the surface in 14 hr with 850 sacks of controlled grout. The bore hole was redrilled again and tested with 925 psi static pump pressure, and the dry test indicated 3 gpm of water in the bore hole, resulting in a decrease of 247 gpm. In this instance, interesting comparisons can be made of the two methods and the advantages of controlled pre-grouting over conventional methods.

No. 1 SE bore hole was again redrilled and seven grout control colored tracers were identified from No. 2 NW. Only 67 pct of No. 1 SE bore hole contained grout which had been previously placed in the conventional manner, and a large portion of it indicated pump erosion. No. 1 SE bore hole was tested and found to be making 97 gpm of water, and this quantity would no doubt have been larger if colored controlled grout tracers had not penetrated the grout hole from No. 2 NW. The bore hole was regrouted with controlled grouting, and comparable packer settings were again used. The operation proceeded in five stages. In each, 50 sacks of special, tailor-made, flash-setting, pre-controlled grout were used, followed by 100 sacks of controlled grout, in which the gravity and viscosity were held within narrow limits to suit individual stage grouting conditions. The control of this operation requires not only a thorough knowledge of the materials used and their behavior, but also, and we cannot stress this too strongly, experience in placing and setting. A total of 750 sacks were placed in the grouted area adjacent to this bore hole. The hole was redrilled following controlled grouting, and tested with the grouting pump at 1000 static psi. The dry test indicated 9 gpm, an 88 gpm reduction.

The drilling and pre-grouting were finished and all results were weighed. The company drilled only 6 grout holes, compared to an established program of 9—a saving of 3 bore holes or $4000.00 to $5000.00. Based upon their previous experience, 10,000 sacks of cement were appropriated for the job while only 6431 were used in controlled grouting; an overall saving in materials and drill time of $8000.00 to $10,000.00. The shaft is now completed and is making 8 gpm of water with one short water ring, the result of heavy blasting. With this exception in the sinking of the shaft, all walls stood firm as reported by the shaft sinking contractor.

Solidifying Mines and Shafts Areas by Pressure Grouting

By B. H. Mott,* Member AIME

Underground water has been one of the greatest problems in sinking mine shafts, sealing existing shafts, and driving headings under streams. In the preparation of a proposed shaft or existing shafts for elimination of water leakage, advances have been brought about through the use of a controlled grouting method.

The modern method to divert water from proposed shaft sites is known to us as pre-grouting, and from existing shafts, as controlled pressure grouting. This method involves the solidification of the surface and rock by the injection of a liquid mixture of

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cement and a certain percentage of rock dust and special clays pumped through a duplex plunger type slush pump that all cavities, fissures, and the entire rock structure in the area be made watertight.

A three-fold advantage is readily found in employing such principles. 1. The elimination of the greater part, or all of the water problem in sinking a new shaft. 2. Sealing-off of leaks in existing shafts without interruption of mine operation. 3. Solidifying of wet mine entries as may be found in driving wet headings under streams.

In the sinking of a shaft, underground water is often a very costly part of its completion. Also, it has been necessary to abandon shafts because of water and unstable ground. Many methods have been devised to combat the difficulties in taking care of a large volume of water under pressure. This problem of sealing off water has not been completely solved, in spite of the numerous plans that have been tried. The freezing method has been found too expensive; as is also true with chemicals. The grouting method has proved the most satisfactory from an economy standpoint.

The filling of cavities and fissures in broken rock formations is not new; however, the old method of grout and grouting has ceased to mean merely a cement mixture pumped into the top of a drill hole and calling it grouted.

It may prove interesting to bring out some experiences in pressure grouting for proposed mine shafts and underground entries. One of the fundamentals is to predetermine at what elevation in the drilled hole grout would be accepted, and at what pressures. In addition, it is necessary to know the volume of water that fissures and cavities would take on a water test. This is done by employing a hydraulic sectional water-testing device that will segregate and test each 5-ft section of the entire hole for water leakage.

Such information is then carefully checked with the drilling log and core itself. The water losses, the inflow, openings, and soft ground, determine the procedure of grouting for that particular hole. If the test shows a large amount of water, rock dust or other filler will be used. As an example, there is used the seven-hole layout for a shaft (Fig 1). This number of holes usually has been found to be sufficient unless the shaft is adjacent to a stream, or difficulties are encountered during the grouting operations. The operation then is as follows: A packer that can be expanded against the wall of the hole for high pressure and released after grout is set, is then placed; beginning at the first grout stop in the bottom of the drill hole, each crevice or fissure is grouted off to the point of no acceptance, or, to the desired pressure which was determined by use of the hydraulic sectional water-testing device.

Where an excessive amount of grout is accepted at low pressure, this is an indication of a direct leak outside of the shaft area. Blocking or building a barrier to keep grout within the shaft area then becomes necessary, since the object is to extend a water-tight plug, or, build a cofferdam around the outside of the shaft. Grouting operations are repeated by moving the packer the proper distance up the hole to the next packer stop or leak and continuing this until the entire hole is completely cemented. (Grout should be bypassed at each packer location holding an even pressure so that all openings are closed and grout will be retained where placed in the hole.)

The method above described is used for existing shafts into which water leaks during the summer and freezes in winter except that very low pressures must be maintained. It is not even safe to state the range of these pressures, due to the fact that some very weak shaft linings may be encountered. Detailed notes are kept of

time, the ratio of cement used, the operating pressure, and possible back pressure from grout which has gone into openings higher in elevation. With these notes it can be determined when to change the mixture or to add certain quantities of swelling compound or chemicals to make a seal or when to bypass the mixture to close a leak.

When a large volume of water is encountered unexpectedly in driving a set of entries under a stream or broken section of ground, pumps and discharge facilities may not be available or of sufficient capacity, hence it may be good economy and less expensive, considering the cost of pumps, power and maintenance over a period of years, to consider grouting in taking care of water.

There have been various methods devised in laying out drill hole locations for such conditions. In this layout of holes, a broken water-bearing slate roof over the coal exists, ranging in thickness from 8 to 15 ft (Fig 2). On this plan, holes in the draw slate on both sides of the entry were drilled on a 15° angle and inclined 15° to the vertical, and one drilled upward in the center in line with the heading on the same inclined degree. The holes are then cored 40 to 50 ft ahead of the working face.

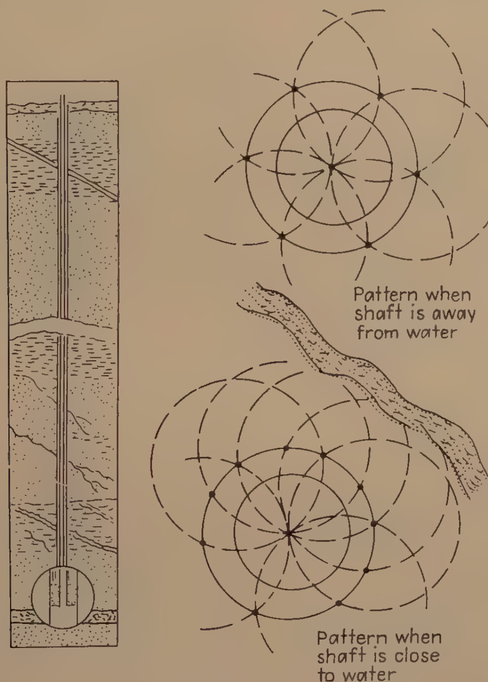

FIG 1*a*—PLAN. DIAMOND DRILL HOLES APPROXIMATELY 15° ANGLE IN DRAW SLATE AND INCLINED 15°.

FIG 1*b*—ELEVATION. DIAMOND DRILL HOLES APPROXIMATELY 15° ANGLE IN ROOF DETERMINED BY CONDITIONS.

Grouting and testing of these angle holes are done by the same method as employed in grouting a shaft site.

Drill hole patterns or locations of holes should be thoroughly studied to dry up a wet entry, or, broken section when necessary to go under a stream, by drilling and grouting from the ground surface. The usual procedure is to drill directly in line

with the heading. One line of holes may be sufficient. We have drilled as many as three lines, all going below the coal seam. This pattern shows a plan developed to give rounding a shaft location or a working area.

There are a large number of grouting jobs at coal and other mines which indicate a

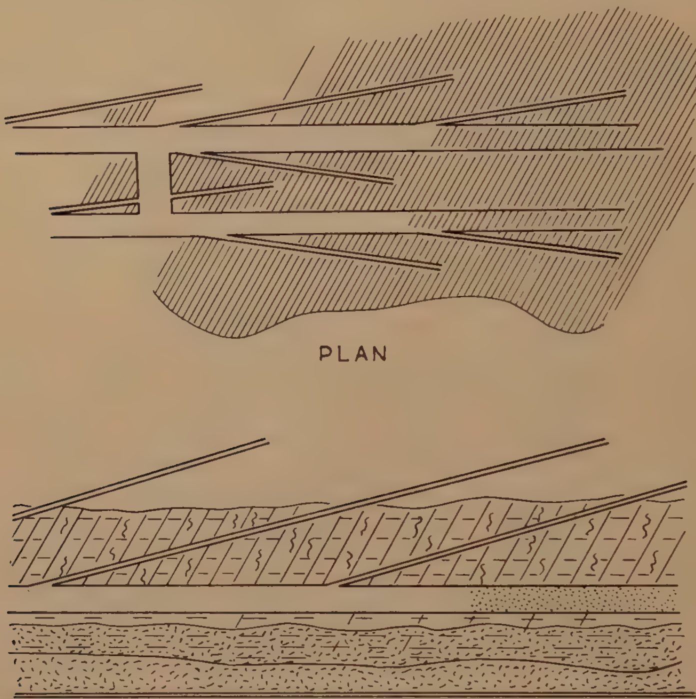

FIG 2—PROPOSED SHAFT DIAMOND DRILL HOLE PATTERN.

some idea of a spread and overlapping for different elevations in the drilled hole. Sealing the sidewalls and the roof does not always make the heading watertight—it may be necessary to grout with short holes in the floor of the mine. While no attempt has been made here to go into detail of this subject, its importance should not be overlooked in mining operations. This is only a preliminary outline of handling a grout mixture to build a watertight barrier sur- strong trend toward this permanent method of sealing water from mine openings and underground passages.

Mining engineers are more interested; shaft contractors are more interested; contractors interviewed are able to reduce their contract price when they can be protected by a water clause of only a few gpm and they can reduce their cost. We have grouted one shaft where it was perfectly dry.

Grouting in Southeast Missouri District

BY W. W. WEIGEL,* ASSOCIATE MEMBER AIME

IN "Mine Drainage, Southeast Missouri Lead District," *Trans.* AIME, (1943) **153**, 74–81, the general water conditions in the lead mines of Southeast Missouri were described. Some comments developed from the experience of time and some further

* St. Joseph Lead Company, Bonne Terre, Missouri.

explanation and additions may now be made to that article.

The original article described how the mines, largely near the base of the 400-ft thick Bonne Terre dolomite underlain by a thick porous sandstone, get only a minor part of the inflow water through the sandstone by artesian circulation. The main intake points for the water handled at these mines are apparently where fracture and channel zones, which in some cases extend from the top to the bottom of the formation, cut under surface drainage lines. These allow large amounts of water to enter the mines either directly when the channel zones are cut by mine workings or else by keeping a slight head on the sand they force water up into the workings. In 1941, the average inflow of the district was 20,000 gpm, which with a 21,200 tons daily production on a six day week, gave a water-ore ratio of 6.5. Opening up of a large amount of virgin territory towards the outer edge of the district has increased this to a pumping load of approximately 25,000 gpm. Production is about the same, but on a five day week basis, so the water-ore ratio is now about 9.5 tons of water for each ton of ore.

Major pumping equipment has changed very little since 1941. Due to the necessity of developing low elevation ore, formerly ignored, the war years saw a considerable increase in the number of relay pumps.

No more areas have been isolated and bulkheaded from the mine as previously described. All previous work has held up well and has been entirely satisfactory. However, the peculiar circumstances of an isolated large wet area of mine workings with connections only through drifts do not often arise. The area mentioned as being shut off by three bulkheads has now been reopened. The plug type bulkheads are merely shot out and the drift is then clear. The lack of heavy reinforcement in the concrete can thus be a distinct advantage if it is desired to remove the bulkhead.

The "Mine Drainage" article described the mass grouting of large areas from the surface using fine flotation tailings spoken of as "slime." This has been done principally along the channel fracture zones and other main sources of water to the mine and has been highly effective. About 450,000 tons have been pumped, directly covering a total area of about 300 acres but affecting much more. During the latter part of the war, because of the stringent pipe situation and the shortage of labor, the program was shut down and has not been restarted. Several new water bearing areas which it would have been advisable to mass grout by "sliming" have opened but these have been too far from the mills (three to five miles). A new shaft recently sunk even farther (eight miles) from an active mill would have been an ideal case. Numerous large channels are in the area and merely sinking of the shaft caused several large sinkholes to form in the area. Several mud runs also occurred during the shaft sinking period and caused much trouble. No mining has been done from this shaft as yet.

During the war years, owing to pressure for production, open stope mining several hundred feet wide was carried across an intensely shattered area which had been one of the first to be mass grouted. Before sliming, even a drift through this had had to be abandoned after excessive water flows. The sliming with some follow-up underground cement grouting had so filled the openings in the area that water production is normally only about 500 gpm from this section.

This "follow-up" grouting work has been important. The sliming does not fill all the spaces and if water flows through these, part of the fine slime will be washed out, making the opening even larger. The grouting procedure in both slimed and unslimed areas is somewhat the same and is given in some detail below.

In drifting through areas where there is probability of water channels, it is cus-

tomary to keep a pilot hole ahead. Long underground diamond drill holes require the shutting down of the drift and frequently have drilled through channels more or less mud-choked without showing evidence of them. Simpler and surer has been the practice of using a 22-ft hole drilled by the regular drift crew each round. By alternating from one side to the other and from top to bottom, these will give warning of the channel even if the angle with the drift is acute. On cutting a water channel with the pilot hole, the water is then shut off with a tapered wooden plug in the hole and preparations made for grouting the channel.

All loose rock is first cleaned off the face and then any small leaks through cracks are caulked up with wooden wedges and oakum. A hole (or several) with a diameter just under 2 in. is drilled into the face for a depth of about 3 ft and a grouting sleeve inserted. These are of 1½-in. pipe threaded on one end for a valve and with a slight taper for about 10 in. on the other end. A thin layer of friction tape is wound around the tapered end and the pipe driven into the hole with a hammer. If the face rock is broken or a very high pressure of water expected, the sleeve may be anchored with ties to anchor eye bolts to the side. A small hole is then drilled ahead of the sleeve with the steel passing through the valve and pipe until the water is cut. After withdrawal of the drill and closing of the valve, leakage of water in the face and around the sleeve may require more calking.

Actual grouting equipment consists of the pressure pumps, mixing barrels and connecting hose and pipe. No. 6 Cameron pumps are used, with an 8-in. air cylinder and with water cylinder bushed down to 3½ in. With 90 lb air pressure this gives approximately 400 lb pressure for grouting. As the maximum hydrostatic head is less than 600 ft, there is ample capacity. Mixing barrels used are round tanks, 36 in. diam by 30 in. high, with a semi-cone bottom. An Ingersoll-Rand No. 25 sump pump is placed in each barrel as an agitator (the discharge of the sump pump is directly back into the barrel). Two barrels are used. Each barrel mix consists of 5 to 10 sacks of cement for about 70 gal of water. While one barrel is pumped the other one has its batch prepared. The entire outfit is mounted on flat cars and should be pushed as close as possible to the face to be grouted; that is, less than 35 ft. Connections are made from the grout pump to the sleeves with high pressure 2-in. pipe and hose with union ends. A blow-off valve is placed in the line as close to the face as possible.

Actual order of grouting varies somewhat. If the channel in the face is thought to have a large amount of thin mud, clear water may be pumped at first to get room for the cement grout. If the face is badly shattered and broken with many leaks, a small amount of very thick grout may be pumped slowly and then allowed to set. Occasionally fine mule feed and a large amount of aquagel may be required as an addition. A new grouting hole is then prepared. It may be necessary to repeat this and gradually build up the rock so that blowouts of grout will not occur. If the face is solid and the channel apparently open, regular pumping is started with a mix of about 5 sacks of cement to 70 gal of water, gradually thickening the successive mixtures up to about 10 sacks per 70 gal until the pump finally stalls.

Common cement only is used. Aquagel is used as an addition of from one to two pounds per sack of cement. This allows much thicker mixtures to be pumped without strain on the pump and the grout sets better to the shape of the channel. The use of any fine sand has been found to be detrimental as there is usually separation in the channel. Temperature of the water is 58°F. Pumping once started is continuous. Four men on a shift have pumped as

high as 300 sacks. As one hole shuts off, another one is started if ready, until the face will not take any more. After the channel is completely grouted, the cement is allowed to set for about a week. The drift is then advanced carefully through the channel until firm rock is met on the other side or more water is hit in a second channel.

The amount of grout pumped into any one channel bears little relation to the initial pressure. Three or four sacks or as high as 1500 sacks may be needed. In one drift that was crossing a channel zone, fifteen channels of various sizes were cut in a distance of about 1000 ft. All had close to 135 lb initial pressure and flows up to 1200 gpm. Grouting averaged about 500 sacks per channel (some up to 1000) and did good sealing. It should be noticed particularly that this area had not been mass grouted by sliming from the surface. The fifteen grouting operations, totaling some 8000 sacks of cement, in a slimed area would be reduced to possibly nine or ten grouting operations and would not use more than several hundred sacks of cement. In slimed areas, initial pressure of channels is quite low, rarely as much as 50 lb, and normally cement grouting will not require more than 30 to 40 sacks at the most. Many of the channels as cut are already completely blocked. The slime grouting and cement grouting are thus seen to be good mutual complements. As the slime has already thoroughly settled and compacted, there is no mixing with the cement grout. The cement fills the remaining small openings and shuts off the last trace of water. The slime grouting cuts the amount of cement grouting to a mere fraction of that required without it. In addition, as it covers a wide area comparatively cheaply, it is not restricted to making a small ring around a narrow opening such as a drift, but is suitable for preparing shattered and fractured areas for large mine openings such as high breast stopes.

DISCUSSION

I. B. Crosby*—The writer recently has had an opportunity to compare European with American grouting methods and he was impressed with the much greater use of chemical grouting in Europe. A few of the typical uses will be mentioned. It is used for tightening finely fractured rock, for producing impervious grout curtains in sand, and for grouting about tunnels in sand. It was used to produce a grout curtain in fractured limestone under the Castillon Dam, now being completed, on the Verdon River, Southern France. It was used also in producing a grout curtain under the existing earth dam at Lac Noir, in the Vosges Mountains, France, where the grout was injected in sand and gravel of a glacial moraine. Experiments and tests have just been completed for a grout curtain more than 300 ft deep in fine alluvium at the proposed Serre-Poncon Dam on the Durance River in Southern France. Mixtures of sodium silicate and sodium aluminate with varying amounts of cement and bentonite were used in each case. An example of tunnel grouting is the work on one of the Paris subways to prevent leakage where it passed through fine sand. A mixture of sodium silicate and sodium aluminate was used and successful results were obtained. Chemical grouting has been used successfully at many other places in Europe for various purposes and also in Algiers, Africa.

The successful use of chemical grouting in other countries raises the question as to whether more consideration should not be given to its use in similar problems in this country.

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Concrete and Wood Blocks for Ground Support in Cyprus Mines

By J. L. Bruce,* Member AIME and G. W. Nicolson†

(Los Angeles Meeting, October 1947; New York Meeting, February 1948)

The country rock of the Mavrovouni mine of the Cyprus Mines Corp. is hydrothermally altered, disintegrated pillow lava, with very little tensile strength ("short" ground). In places, especially when wet, it is heavy ground, though not extremely heavy except where movement has been caused by underlying or nearby stoping operations. Damage by caving from roof or sides of galleries is not sudden but progressively faster when the size of the cave increases as a result of the fall of small pieces.

Cement and Concrete Linings

During early operations, galleries in the country rock were supported by customary mine-timbering methods. Available timber, however, was of poor quality and of relatively high price, and replacements were frequent and expensive. Furthermore, the massive sulphide ore bodies required large volumes of fresh air for ventilation and cooling, and it was therefore desirable to provide smooth-lined galleries with low resistance for the introduction of air to the ore bodies. It was also desirable to eliminate hazards of fire in shafts and underground workings insofar as this would be practical. Cement and concrete were relatively cheap, and plans for using these in support of shafts, underground galleries, raises, and chutes, were developed steadily. Circular shafts with diameters ranging from 10 to 15 ft were adopted for all permanent shafts and airways. In places these were lined with concrete poured behind sectional pressed-steel forms especially constructed for this purpose. In other places, they were lined with precast-concrete blocks. It was found that such precast concrete of good quality could be produced on mass production scale and delivered at the mine from most favorable point of manufacture, at a cost of about 25 cents per cubic foot of concrete. This refers to prewar costs.

As the use of concrete blocks in circular lined galleries was developed, it was found that there were very great advantages in the reduction of air resistance as compared with customary timber supports. After best methods had been developed, it was found that breakage and replacements were extraordinarily low, averaging only about 1½ pct per year during a period of about 15 years. Furthermore, it has been possible to reclaim for re-use a very large percentage of the concrete blocks from abandoned galleries. During the history of the operations precast-concrete blocks have been used for lining shafts, raises, ore chutes, galleries, and shaft stations, ranging from 3 to 16 ft id. The thickness of lining has varied from 5 to 10 in.; with 12 in. thickness in a few cases.

Junctions of two galleries at angles ranging from about 30 to 60° are usually made of poured concrete behind sectional forms, or of a combination of poured concrete and precast blocks. In nearly all cases it has been found advisable to make the point of the wedge at the junction, of

Manuscript received at the office of the Institute Oct. 24, 1947. Issued as TP 2413 in Mining Technology, July 1948.

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poured concrete, and it has been found convenient to make this in the form of a concrete post, 24 to 30 in. in diameter, using oil drums, carbide drums, or similar required large stocks on hand of many different blocks. These blocks were all segments of rings up to 3½ ft long and about 6 by 6 in. cross section. In handling

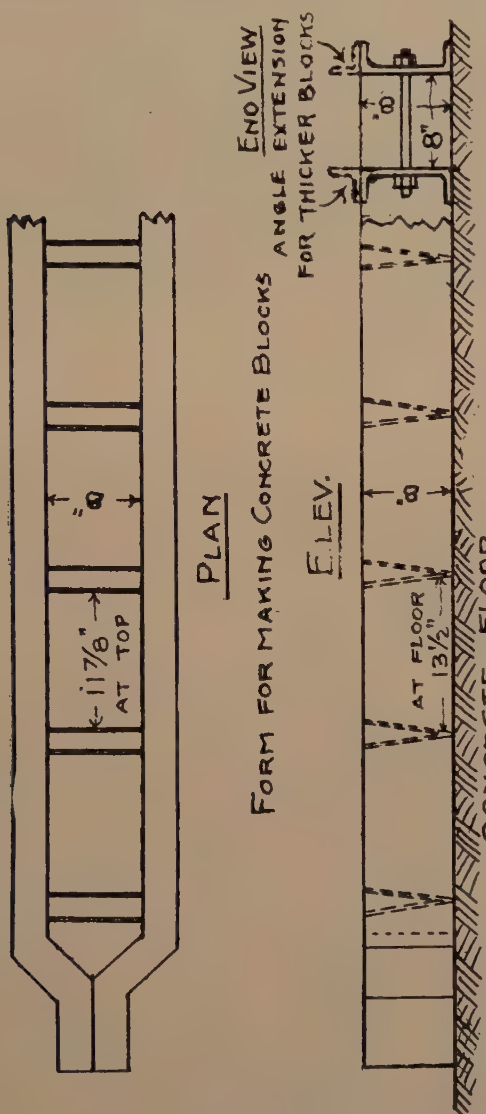
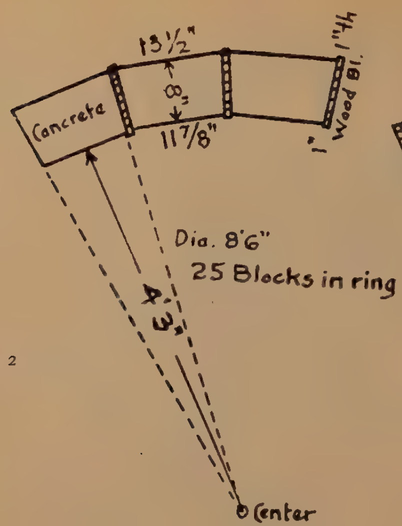


FIG 1—PLAN AND ELEVATION OF STANDARD FORMS FOR MAKING CONCRETE BLOCKS.

discarded drums, as forms, making sure that the base of the column is placed on solid ground.

Originally, precast blocks were made in many shapes and sizes and consequently these blocks before concrete was set, and after placement in galleries, the breakage was considerable. The long blocks contained four ¼ in. round iron rods for reinforcement which was more necessary while



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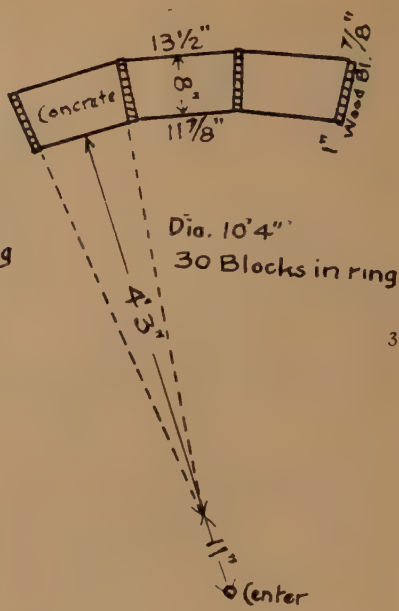

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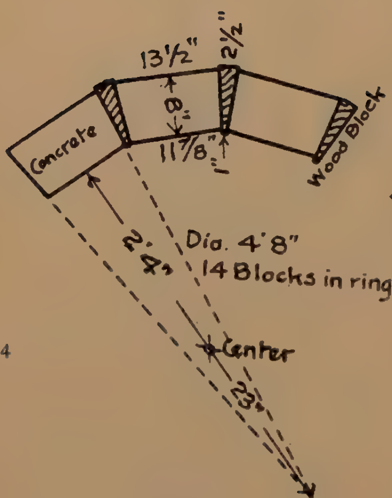

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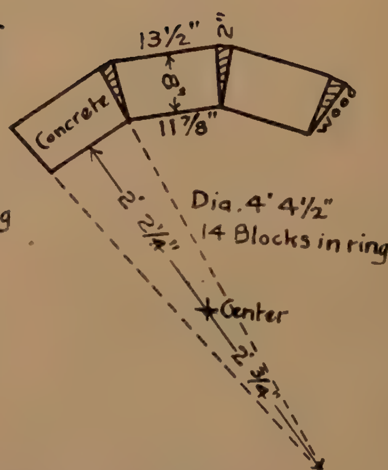

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FIG 2—LAYOUT OF CONCRETE BLOCKS FOR 8 FT 6 IN. DIAM GALLERY.
FIG 3—LAYOUT OF CONCRETE BLOCKS FOR 10 FT 4 IN. DIAM GALLERY.
FIG 4—LAYOUT OF CONCRETE BLOCKS FOR 4 FT 8 IN. DIAM GALLERY.
FIG 5—LAYOUT OF CONCRETE BLOCKS FOR 4 FT 4½ IN. DIAM GALLERY.

FIG 6—SHOWING METHOD OF ADVANCING GALLERY HEADING BEFORE PLACING BLOCK-LINING GROUND SUPPORTS.

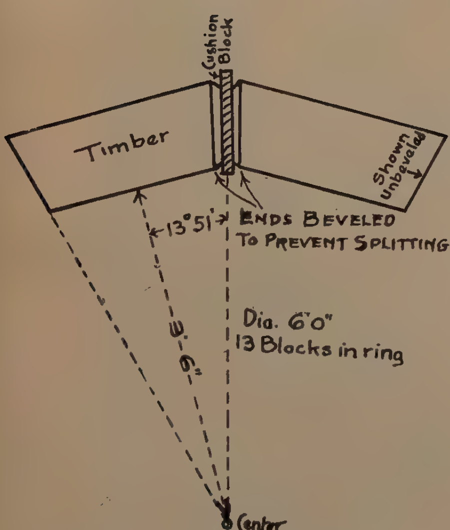

FIG 7—LAYOUT OF TIMBER BLOCKS FOR 6 FT 9 IN. DIAM GALLERY.

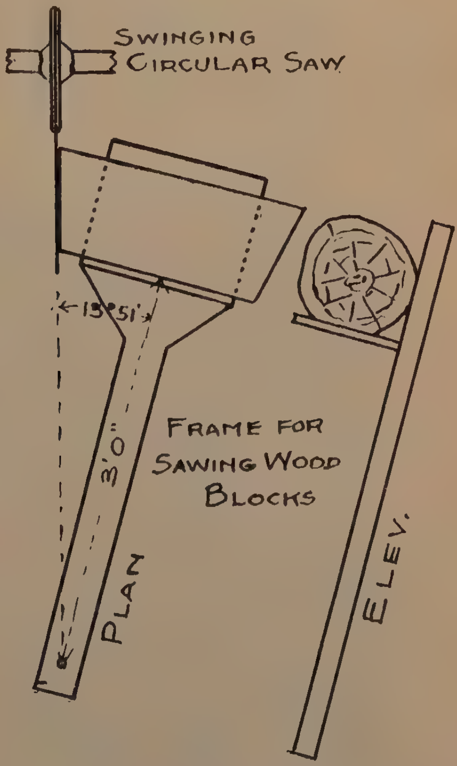

FIG 8—PLAN AND ELEVATION OF FRAME FOR SAWING WOOD BLOCKS.

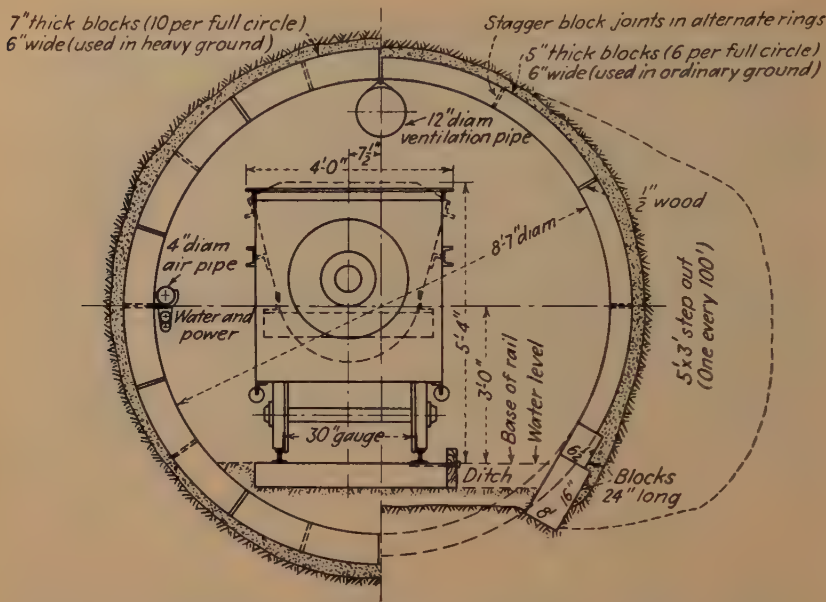

FIG 9—CROSS-SECTION AND PROJECTION OF CONCRETE-BLOCK-LINED GALLERY.

Left half shows standard construction for heavy ground. Right half shows alternative lighter construction for moderate ground with poured concrete base and incomplete circle. Position of "rocker-dump" car is shown by broken lines superimposed over standard box car.

FIG 10—CONCRETE-BLOCK-LINED GALLERY 8 FT 3 IN. ID, 30 IN. TRACK GAUGE, 25 BLOCKS PER RING. Drain ditch on left side of track. Wood compression blocks show between concrete blocks.

handling blocks when still "green," than during service in the mine.

The recent practice is to make one or two standard sizes and one shape of concrete inconveniently heavy for handling in the mine.

These blocks are made in steel forms bolted together, as shown in Fig 1. A

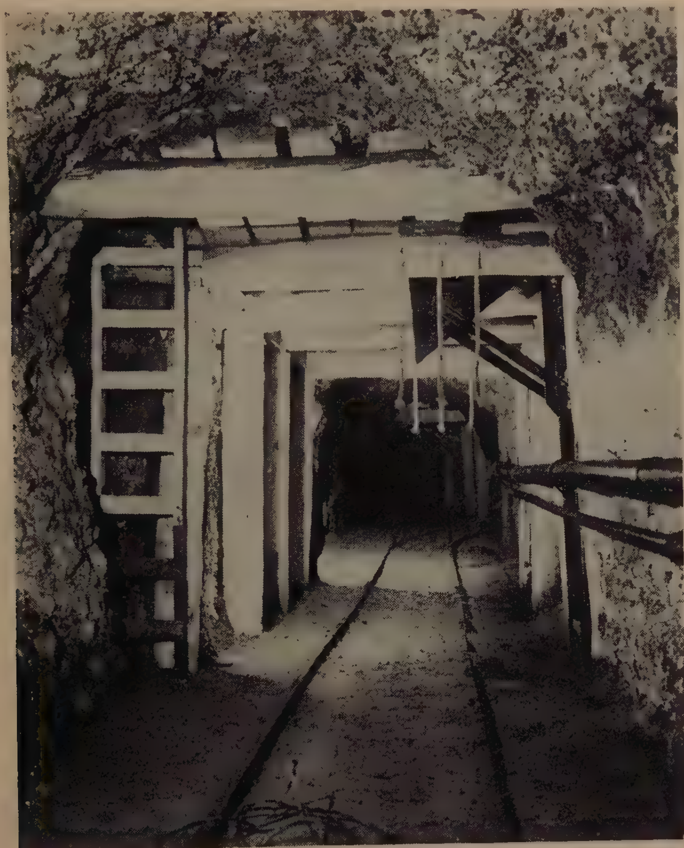

FIG 11—HAULAGE DRIFT IN HARD ORE.

Shows standard steel loading chute mounted on timber frame, and timber-drift sets to support the chute operator's platform, also ladderway to the platform and to the manway alongside chute. Swinging ropes to warn the train crew of the chute lip. Whitewashed for better illumination.

blocks without reinforcing iron and to use wood cushions between the ends of all concrete blocks.

The present standard block is 8 in. square and 13½ in. on the longest side and 11⅞ in. on the shortest side, without curvature.

This block contains 0.47 cu ft and weighs about 70 lb. One cubic yard of concrete will make 57.3 blocks. Larger blocks are tough brown paper is placed on oiled concrete floor. The forms are assembled, bolted together and placed on top of the paper. The concrete is tamped into the form and allowed to stand 6 to 12 hr when the form can be stripped off the blocks and reset for making additional blocks, leaving the concrete blocks on the floor for a day or two longer before stacking them outside for final curing. After 30 to 60 days the blocks

are ready to go underground. When the forms are filled, the date is marked on each block with a pointed tool. This dating is advisable to ensure that the blocks will be are 1 in. thick on the inside and ⅞ in. thick on the outside.

The wood cushion blocks are used to distribute pressure of the adjoining ends of

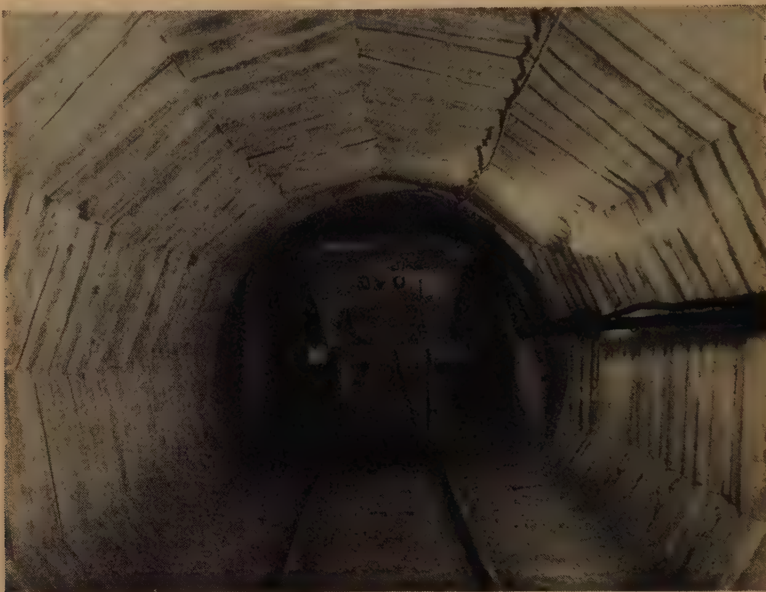

FIG 12—WOOD-BLOCK LINING FOR HEAVY GROUND.
This gallery is 8 ft id using 13 blocks per ring. The track gauge is 30 in. The car shown is 3½ ton (long) rocker dump type. Blocks have been in place 4½ years.

properly cured before using them. Quick setting cement should be used when it is necessary to place blocks soon after they are made.

It will be apparent that many forms will be required for making precast-concrete blocks on a "mass production" basis. Concrete mixing machines, pneumatic tampers, compressed air, and good plant layout for moving materials and blocks are also advisable.

Fig 2 shows the standard concrete block with 1-in. parallel wood block for cushion as used in 8 ft 6 in. diam drift or gallery. It requires 25 concrete blocks and 25 wood blocks to complete the ring for 8 in. advance of the lining. Fig 3 shows the same standard block used in lining a 10 ft 4 in. diam tunnel, of which considerable lengths have been driven. The wood cushion blocks concrete blocks. It has been found that this very greatly reduces breakage of the corners of the concrete blocks and greatly increases strength of the rings. The wood blocks may be treated with Woolman salts, cupreous mine waters, or other chemicals, to prevent decay and resist fire.

In placing these block linings in underground drifts or galleries, a small pilot heading with customary timber sets—or untimbered—is kept 20 ft in advance of the lining. This gives indications of the conditions ahead and any change in ground—or water—is seen before lining with concrete (see Fig 6).

The grade and alignment is given by the engineers. A radial measuring rod is used in measuring from a string or wire center line when enlarging a gallery for setting the

blocks. Generally 2 or 3 ft are excavated ahead of the lining. Three rings are started in the floor with the joints staggered. After the blocks in the lower half of the three rings are in place, a curved form supported on a jack screw is used to support the upper half of one to three rings until the key blocks are in place. This circular support is set 3 in. high to allow ample space to get all the blocks and wood spacers in place. Then the screw is lowered and the blocks wedged down into place. Great care is taken to back fill any cavities in the roof or sides outside the block lining as the strength of the ring of blocks depends on its circular shape and any cavity that is not filled might allow the blocks to get out of shape and weaken the ring.

Vertical manways and chutes are lined as shown in Fig 4 and are 4 ft 8 in. id using 14 concrete blocks with wooden wedges 1 by 2½ in. size. The manway and chutes are generally vertical.

The 4 ft 8 in. id concrete-block chutes may be wood lined to prevent wear on the concrete blocks in places where large tonnages must pass through chutes. The wood-block ring is 30 in. id and 40 in. od and consists of 18 blocks 10 in. long and 6, 7 or 8 in. high, depending on the size of timber available. The wear of the ore is taken on the end grain of the wood. The space between the concrete block and the wood blocks is packed tightly with moist clay. The timber used was eucalyptus, which, when dried, is hard, tough wood.

Fig 5 shows a special manway lining used in a fire area. The pointed wedges leave no wood exposed to fire risk.

If the ground is very heavy so that 8 in. thickness of concrete blocks is insufficient, the thickness may be increased to 10 or

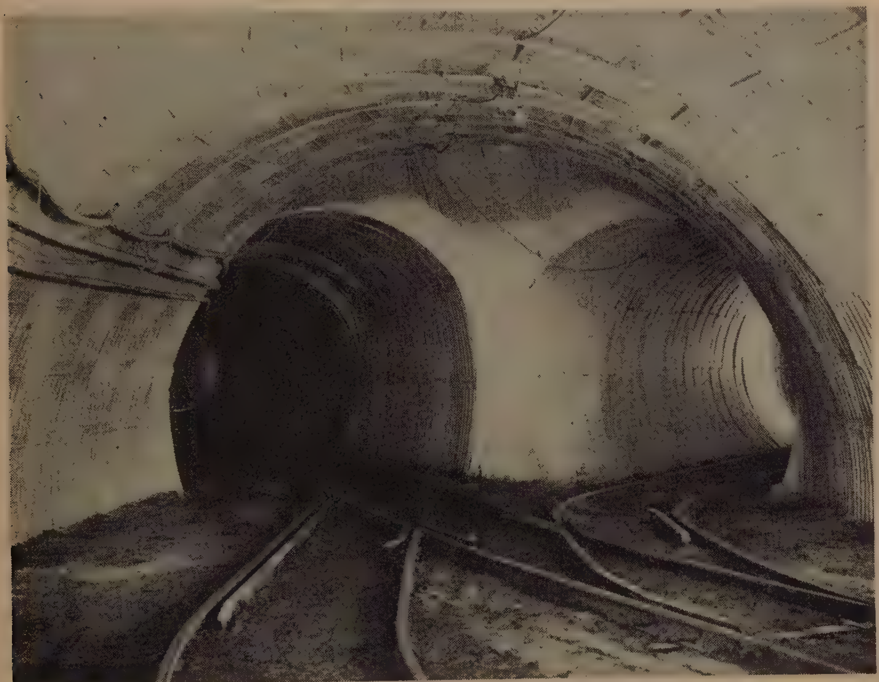

FIG 13—A POURED CONCRETE-LINED STATION 16 FT WIDE BRANCHING INTO 8 FT DIAM GALLERY STRAIGHT AHEAD, AND 8 FT DIAM GALLERY ON CURVE.

12 in. if special heavy precast-concrete blocks have been made, seasoned and stocked ready for use when required. These blocks can be made in the same steel form by building up the depth of the steel form with angles bolted to the channel members as indicated in the end elevation shown in Fig 1 if the spacers between blocks have been made to extend to the top of these angle-iron extensions. This makes a smaller dimension for the short side of the blocks and requires about one more block for each ring if it is desired to maintain the same interior clearance.

Fig 14—A 16 ft poured concrete-lined station reduced to 8 ft diam by holding left wall straight and reducing the diameter 1 ft for each $1\frac{1}{8}$ ft advanced.

Wood Block Lining

Fig 7 shows a wood segment lining for circular tunnel such as was used in "*moving*" ground where concrete blocks failed.

The wood lining is 6 ft id using 13 segments. The odd number of segments allows the joint to be on the center top with a flat face on the floor to accommodate the track. No lagging is used. The round timbers are placed side by side (skin to skin). The timbers—10 to 12 in. in diameter—are slabbed off on one side before framing. This flat face is used in the saw mill to get the ends cut in the proper position and is a great help underground in lining up the sets. Blocks of approximately the same diameter are selected for use together in any one ring of lining.

Fig 8 shows the frame used in the saw mill to cut these segment blocks.

Fig 9 illustrates alternative designs for ground support in tramming galleries.

Wood-block lining will be more durable if the ends of blocks are beveled off and wood cushion blocks used as indicated in Fig 7, as this imbeds the grain of the timber in the cushion block and tends to pre-

vent splitting. When using wood blocks for tunnel lining in heavy ground it is advisable to use in each ring a sufficient number of blocks of such diameter that the length of the short side of any block will be not more than 2.5 times the diameter of the timber used in the block. For example, in lining a circular gallery with internal diameter of 6 ft it is advisable to use in each ring 8 (or more) blocks if 12 in. timber is used while there should be about 12 blocks if lining is to be made of blocks with diameter of only 8 in. If inside diameter of gallery is to be 8 ft there should be 8 blocks per ring (octagonal) if diameter of timbers is about 17 to 18 in.; and about 12 blocks (dodecahedral) if timbers are only 12 in. diam.

When it is desired to make a curved turn in a gallery the block linings are set in groups of 3 rings each with each group touching the next group at the inner side of the curve. A space is left between groups of 3 rings at the outer side of the curve so that the center line of the middle ring remains always approximately normal to the curved center line of the gallery. The space to be left between groups (of 3 rings each) at the outer side of the curve is calculated by the engineers and given to the shift boss in charge of driving and lining. Spaces between groups should be filled with poured concrete or timber blocking. Fig 10 to 14 show several types of lined galleries.

A Review of Rock Pressure Problems

By Richard P. Schoemaker,* Student Associate AIME

Introduction

In underground mining operations the effects of economic and mechanical factors on costs and profits can readily be appreciated and can perhaps be expressed in exact figures and percentages. Less apparent, but frequently more important, is the influence of physical stress.

Stress causes failure of supports, collapse of openings, rockbursts, and other damage which directly affects the costs of maintenance, and also to a certain extent the accident rate. Stress can, however, if properly directed, be an aid to mining operations. The caving methods in metal mining and the long-wall method in coal mining are well-known examples.

In large-scale mining operations it is almost impossible to neglect stress-control. A better understanding of the mechanism of rock pressure and the application of its principles to practical mining will inevitably lead to a reduction in operating costs.

Many theories have been advanced to give the mining engineer a practical outline for stress-control but, although today most experts agree upon the principles of stress distribution around underground openings, there is still no generally accepted theory that explains all phenomena. The main reasons for this incomplete understanding of the problem are the lack of accurate knowledge of the physical properties of rocks under field conditions, and the great complexity of these conditions due to inhomogeneity of the rock, stratification, tectonic disturbances, and many other unknown factors.

The research on the rock pressure problems has, however, yielded some valuable facts which, expressed in practical terms, and adapted to local conditions, may become a valuable aid for mining operations.

The purpose of this paper is to give a review of the most important accomplishments of this research with respect to the stress phenomena around small underground openings, such as tunnels and shafts, so as to arrive at an outline of the problem as it stands today.

Special emphasis is placed on a new theory by the German mining engineer, Rudolf Fenner, which has never before been published or discussed in English.

The Pressure Dome Theory

The formation of a dome-shaped space around underground openings is well known from field observations. When openings are made in rock under stress, the rock in the top of the openings will often fail, and caving will continue until the openings have assumed a dome-shaped form, after which equilibrium apparently is re-established.

The oldest theory which attempted to explain the phenomenon is the "beam" theory, which postulates that the hanging wall consists of stratified measures that behave as independent beams with fixed ends and a uniformly distributed load. When failure of the beam occurs, inward shearing at the ends will cause the formation of a dome-shaped space.

The pressure dome theory, which more or less evolved from the beam theory, merely

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Undergraduate prize paper of the Sixth National Student Prize Paper Contest, awarded at the annual meeting, February 1948. Manuscript received at the office of the Institute Feb. 14, 1948. Issued as TP 2495 in Mining Technology, November 1948.

* Formerly Student, Colorado School of Mines; now with Lake Shore Mines, Kirkland Lake, Ont., Canada.

postulates the formation of a pressure ring or dome around the excavation. The rock within the dome fails under its own weight, expansion, and perhaps also inherent stresses. If the hanging wall is not allowed to cave freely, it will throw its full weight on the supports of the opening.

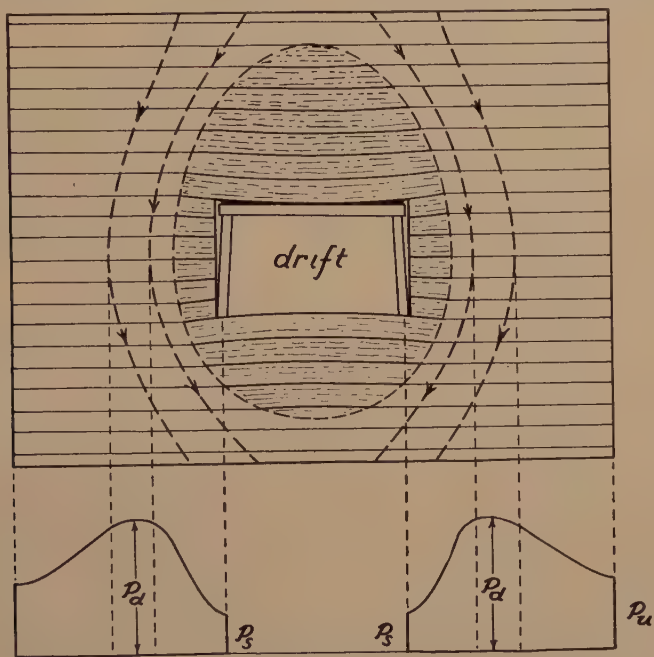

FIG 1—PRESSURE DOME AND STRESS TRAJECTORIES AROUND A DRIFT (*After Dinsdale*[1]).

The theory has been presented in many variations, and many different shapes of the pressure dome have been suggested, but all are based on the principle that the cause of rock failure lies within the dome (gravity, expansion, inherent stresses).

As an example a brief outline is given of the ideas advanced by Dinsdale.[1] Fig 1 represents the cross section of a drift; the egg-shaped curve is the pressure ring according to Dinsdale. The hanging wall within the ring is separated from the rock outside the ring by shearing and rests upon the supports of the drift. The lack of symmetry between the parts of the curve in hanging wall and footwall is due to the influence of gravity.

The pressure profile through the drift shows that the pressure immediately to the sides of the drift (P_d) is considerably larger than the pressure in the undisturbed rock (P_u), because the sides or "dome supports" also have to carry the weight of the rock column above the dome. The pressure on the supports in the drift (P_s) is comparatively small, since the supports have to carry only the weight of the rock inside the dome.

The pressure on the supports is of immediate importance to the miner and it is therefore important to know the relation between this pressure, that is, the height of the dome or the weight of rock to be supported, and the thickness of the overlying strata.

The increase of pressure on the supports is proportional to the increase in depth, according to Dinsdale, and he proves this statement with a simple static analysis

[1] References are at the end of the paper.

(Fig 2). In this analysis the following symbols are used:

H = horizontal component of the pressure in the sides of the drift.

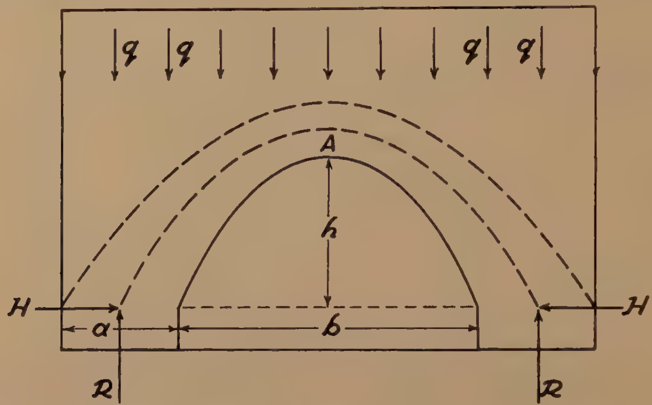

FIG 2—FORCES ACTING ON PRESSURE DOME (*After Dinsdale*[1]).

R = vertical component.

q = pressure exerted by the overlying rocks.

b = width of the drift.

a = width over which the load of the rock column above the dome is distributed on either side of the drift.

h = height of the dome.

It is assumed that $R = q\dfrac{2a + b}{2}$.

The equation of the moment around point A in the top of the dome (considering only the forces to either the left or the right of A) is:

$$M_a = \frac{q}{2}\left(\frac{2a+b}{2}\right)\left(\frac{2a+b}{2}\right) + h \cdot H - R\left(\frac{a+b}{2}\right) = 0$$

Substituting R gives: $h = \dfrac{q}{8H}(2ab + b^2)$ $= (\text{constant}) \times \dfrac{q}{H}$, or the height of the dome is directly proportional to the vertical pressure and inversely proportional to the horizontal reaction.

It is doubtful whether a simple representation of the forces involved, as is given in Fig 2, is permissible. The value of the proof is also doubtful because the relation between q and H is not known. If q and H have a fixed ratio, the height of the dome (h) would remain constant regardless of the depth, as is found to be true according to the theoretical analysis discussed later.

A general criticism on the pressure dome theory may be summarized in the following statements:

1. The formation of a dome-shaped space around underground openings is known from field observations. In the pressure dome theory this is accepted as a fact, and no adequate explanation is given for the occurrence of the pressure dome.

2. The theory postulates (without proof) that the origin of the forces, which cause rock failure in the dome, lies within the dome and not outside.

3. It is doubtful whether the magnitude of the acting forces (gravity, expansion, inherent stresses) assumed by the theory is sufficient to break up the rock within the dome, especially if the rock is homogeneous and has no planes of weakness.

4. The theory as formulated holds only for depths at which the state of stress of the rock remains below the elastic limit.

Despite the limitations and the many questions the theory leaves unanswered, the pressure dome theory is still favored in

many countries. Recent research, however, has developed a completely new concept of the rock pressure problem, as will be discussed in the next section.

Results of Photoelastic Stress Analysis

The photoelastic method of stress analysis is relatively new, but it has been used extensively in the design of machinery and

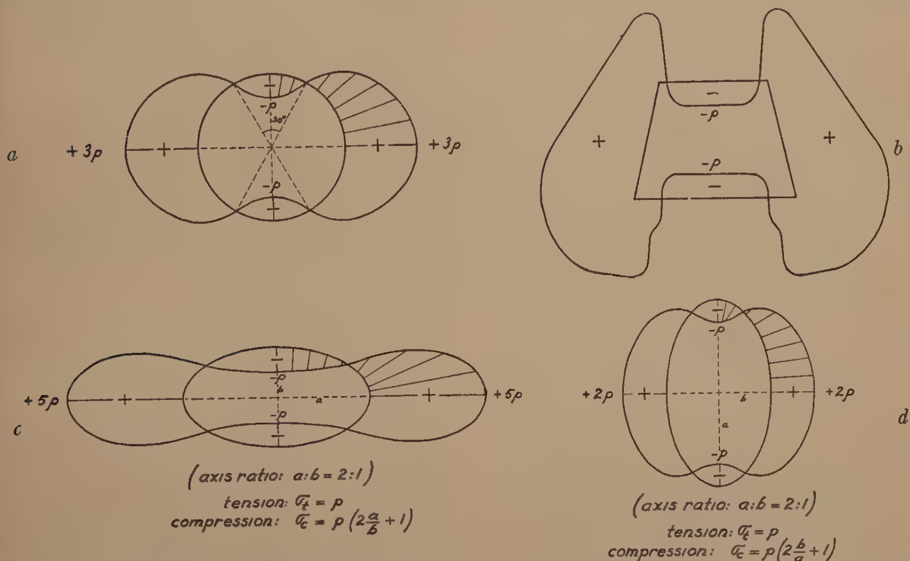

FIG 3—STRESS DISTRIBUTION AROUND OPENINGS OF DIFFERENT SHAPES UNDER VERTICAL STRESS, DETERMINED BY THE PHOTOELASTIC METHOD.
a. Tangential stress around a circular opening (*adapted from Kafadar*[3]).
b. Tangential stress around a trapezoidal opening (*after Dorstewitz*[2]).
c. Tangential stress around an elliptical opening (*after Kafadar*[3]).
d. Tangential stress around an elliptical opening (*after Dorstewitz*[2]).

THE NEW CONCEPT

The research on the rock pressure problem has advanced rapidly in the last ten years, and the results indicate that research is well beyond the stage of trial and error. Although there are still many problems to be solved, we have at present a fair knowledge of the basic factors that govern the behavior of rocks under stress.

The important fields of the present research are; (1) experimental stress analysis by means of the photoelastic method, and (2) theoretical stress analysis in which the solution of the problem is approached with the aid not only of the laws of elasticity but also of the laws of plasticity.

Both methods of investigation give an adequate explanation of most of the phenomena observed in the field.

mechanical equipment in which stresses are important factors. It has also been applied successfully to the study of the rock pressure problem.

The method is based upon the fact that stress causes a change of the optical properties of transparent isotropic material that is visible under polarized light as a pattern of light and dark fringes or color bands. From this pattern, the direction, magnitude, and distribution of the stresses can be determined.

The stresses at the boundaries of small underground openings are of primary importance since they determine the strength of the supports which have to be used.

Fig 3 shows the distribution and magnitude of the tangential stress around openings of different shapes, determined by the

photoelastic method. The test specimens were subjected to a vertical load acting perpendicular to the axis of the opening. The value p is the stress applied or the load divided by the horizontal cross-sectional area of the specimen outside the opening.

The experiments show that the tensile stress at the boundary never exceeds the primary stress p, regardless of the shape of the opening. The trapezoidal opening[2] has the most unfavorable shape since it gives rise to considerable stress accumulations at the corners. In the circular opening[3] the vertical pressure p induces a compressive stress of $3p$ at the ends of the horizontal axis. The magnitude of the maximum compressive stress in the elliptical openings[4,5] depends upon the axis ratio. Mathematical analysis shows that the maximum compressive stress is equal to $p\left(2\frac{a}{b}+1\right)$ if the direction of the applied load is parallel to the minor axis b, and equal to $p\left(2\frac{b}{a}+1\right)$ if the direction of the applied load is parallel to the major axis a. For the circle, where $a = b$, the compressive stress becomes $3p$, as previously stated.

Tensile stress is usually the first cause of rock failure since the strength in tension of all rocks is considerably less than the strength in compression. The occurrence of tensile stress in the top and bottom of the openings, shown by the model experiments, explains why rock failure so frequently occurs first in the top of a tunnel. Failure also occurs in the bottom but is less apparent since gravity tends to hold the rock in place. The experiments also show that the origin of the stresses that cause failure lies in the applied load and not in the pull of gravity, as the conventional pressure dome theory postulates.

The photoelastic model experiments do not, however, give a true picture of what actually happens when an excavation is made in rock under stress, since only the effects of the vertical stress are shown. An accurate analysis of the phenomenon can be obtained only if the horizontal stress is also taken into consideration, as will be discussed.

Results of Mathematical Stress Analysis

In the past, many theories have been advanced to explain the rock pressure phenomenon, but they all proved to be inadequate or applicable only under very special conditions, because they were based upon an interpretation of field observations rather than upon mathematical considerations.

Two interesting new theories will be discussed which have, unlike other theories, a sound mathematical basis.

The first theory was advanced by the German mining engineer Fenner,[6] and the second by Van Iterson,[7,8] director-general of the State Coal Mines in Holland.

Fenner's Theory—Fenner's analysis originated from the conclusion that all attempts to express the rock pressure phenomenon mathematically hitherto failed because they were all based upon the general validity of Hooke's law and also because equations and formulas were used that postulated approximate equal strength in compression and in tension of the material under consideration, whereas the tensile strength of rocks is generally much smaller than the compressive strength.

Fenner's analysis shows that in rocks under pressure there is a certain limit beyond which Hooke's law loses its validity, and the theory of elasticity must be replaced by a theory of plasticity so that the theoretical conclusions will not contradict the field observations.

In order to arrive at the basic factors that govern the stress in rocks, Fenner first analyzed the stress in an ideal homogeneous medium. The cause of the stress in any point of such an ideal rock lies in the weight of the overlying mass, and the relation between the three principal stresses at a point in the undisturbed rock is determined by

the fact that horizontal expansion, induced by the vertical load, is prevented by the surrounding rock.

The state of stress in any point can be represented by the three principal stresses: σ_z along the vertical Z axis, σ_x and σ_y along the horizontal X and Y axis.

The magnitude of the vertical stress can be expressed in the equation: $\sigma_z = \gamma h = p$, in which γ represents the specific gravity of the rock and h the depth below the surface. We also know that the strains in the X and Y directions are zero. The solution of the equations for the principal stresses and strains gives us the following relation between the vertical and horizontal stresses: $\sigma_z = p, \sigma_x = \sigma_y = \frac{p}{m - 1}$; m is Poisson's ratio, which is the ratio between the strain or deformation in the direction of the stress and the strain perpendicular to this direction.*

The state of stress at a point can thus be represented by an ellipsoid with the major axis (axis of rotation) equal to p and the minor axis equal to $\frac{p}{m - 1}$.

The magnitude of the shear stress in any direction can be represented by Mohr's circle for a two-dimensional state of stress, since the horizontal stress is the same in all directions. The maximum shear stress will occur in a plane which makes an angle of 45° with the horizontal and the rock will be in an elastic state as long as this stress does not exceed the ultimate shearing strength of the rock.

When an opening is made in the elastic zone of the rock, the original state of stress will be disturbed and the rock will have the tendency to fill the space. Movement towards the opening will, however, be prevented so that the new state of stress can also be expressed mathematically.[9]

The distribution of the tangential stress around a horizontal cylindrical tunnel, determined by Fenner, is shown in Fig 4*a*. The magnitude of the stress at each point of the boundary is indicated by the length of the line perpendicular to the tangent at the boundary. In the roof and the floor a tensile stress exists equal to $0.25\ p$. The maximum compressive stress is equal to $2.75\ p$.

The value of Poisson's ratio can for most rocks be taken as 5, so that the horizontal stress will have a magnitude of $0.25\ p$.

A stress distribution as shown in Fig 4*a* could also be obtained if we superimpose upon the stress diagram of Fig 3*a*, a horizontal stress equal to $0.25\ p$. The tensile stress at the top and bottom would then become: $p - 3(0.25\ p) = 0.25\ p$, and the compressive stress at the sides: $3p - 0.25\ p = 2.75\ p$.

The same procedure can be followed for the openings with elliptical cross section.

If the horizontal stress is taken as $\frac{p}{m - 1}$ we can arrive at the following general formulas for the stress at the vertical and horizontal axis of any ellipse with the major axis parallel to the vertical stress:

$$\text{vertical}\quad : \frac{p}{m - 1}\left(2\frac{a}{b} + 1\right) - p \quad \text{(tension)}$$

$$\text{horizontal}: p\left(2\frac{b}{a} + 1\right) - \frac{p}{m - 1} \quad \text{(compression)}$$

in which

a is the major axis,
b the minor axis of the ellipse.

The tensile stress becomes zero if $\frac{p}{m - 1}\left(2\frac{a}{b} + 1\right) - p = 0$, or if $a{:}b = (m - 2){:}2$.

For Poisson's ratio equal to 5 the axis ratio becomes: $a{:}b = 3{:}2$, as is shown in Fig 4*b*.

The stress distribution for an axis ratio

* The European usage of Poisson's ratio (m) as the "ratio between the strain or deformation in the direction of the stress and strain perpendicular to this direction" results in the value of the ratio being greater than one; while in American literature, Poisson's ratio is expressed as a reciprocal of the above so that the value never exceeds one.

$a:b = 2:1$ ($m = 5$) is shown in Fig 4*c*, for comparison with the stress distribution around an ellipse with the same axis ratio, determined by the photoelastic method (Fig 3*d*).

tensile strength of the rock. The rock in the roof will fail and caving will continue until the tensile stress becomes equal to the tensile strength of the rock. Thus the opening will have the tendency to attain the form

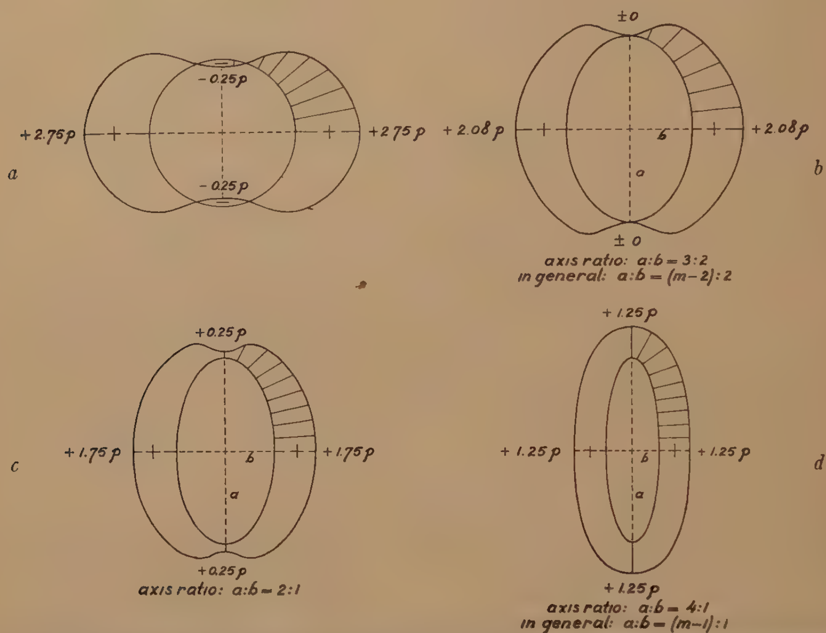

FIG 4—STRESS DISTRIBUTION AROUND A CIRCULAR OPENING AND AN ELLIPTICAL OPENING UNDER VERTICAL AND HORIZONTAL STRESS, DETERMINED MATHEMATICALLY.

a. Tangential stress around a circular opening (*adapted from Fenner*[6]).
b. Tangential stress around an elliptical opening.
c. Tangential stress around an elliptical opening.
d. Tangential stress around an elliptical opening (*adapted from Fenner*[6]).

The stress distribution is ideal when the tangential stress is uniform in all points of the boundary, as is shown in Fig 4*d*. This condition exists when $\frac{p}{m-1}\left(2\frac{a}{b}+1\right) - p = p\left(2\frac{b}{a}+1\right) - \frac{p}{m-1}$, or when the axis ratio is $a:b = (m-1):1$.

Fenner's original idea to determine the magnitude of the horizontal stress and to apply this stress in his calculations throws a new light on the pressure dome phenomenon.

The tensile stress of $0.25\,p$ in the roof of a cylindrical tunnel will usually exceed the tensile strength of the rock. [sic: continues in right column above]

shown in Fig 4*b*, commonly known as the pressure ring or dome.

Fenner's analysis also proves that the height of the pressure dome must be independent of the depth below the surface, since the axis ratio of the ellipse in which the tensile stress becomes zero depends only upon Poisson's ratio.

The axis ratio of the "pressure ellipse" around underground openings can be determined if Poisson's ratio for the rock in and around the ellipse is known. The form of the ellipse will, however, deviate from the theoretical form if inherent stresses are present, or if part of the rock behaves as a

plastic mass, as often occurs in sedimentary rocks.

In the foregoing discussion it was assumed that the maximum shear stress in the undisturbed rock never exceeded the shear strength of the rock. The maximum shear stress is equal to $\frac{\sigma_z - \sigma_x}{2}$; $\sigma_z = p$, and $\sigma_x = \frac{p}{m-1}$; this can be written: $\tau = \frac{p(m-2)}{4(m-1)}$. If $m = 5$ and $p = \gamma h$, the shear stress is $\tau = \frac{3}{8}\gamma h$.

If the shear strength of the rock is exceeded, initial failure will occur and the state of stress will change radically. The formulas of the theory of elasticity no longer hold and the relation between the stresses depends only upon the internal friction of the rock.

The depth at which the elastic state of stress (elastic zone) passes into the plastic state of stress (plastic zone) can be calculated for different rocks as follows:

1. Plutonic rocks: shear strength: 2800 psi
   specific gravity: $\gamma = 2.7$
   Poisson's ratio: 5

The depth at which the shear strength will be exceeded can be calculated from: $\tau = 2800 = 2.7\,\frac{62.4}{144}\,h\,\frac{3}{8}$, or $h = 6500$ ft. This depth is greater than the average operating depth today. In general only the elastic state of stress need thus be considered for plutonic rocks.

2. Sandstone: shear strength: 570 psi
   specific gravity: $\gamma = 2$
   Poisson's ratio: 5

The equation for the shear stress gives us: $\tau = 570 = 2\,\frac{62.4}{144}\,h\,\frac{3}{8}$. The shear strength will be reached at a depth of 1750 ft, which is well within the depth range of present mining operations.

The depth of the upper limit of the plastic zone will in general be less than the theoretical depth, since the strength of the rocks is usually weakened by cleavage and cracks.

In metal mines, where the rocks are usually of plutonic origin, the consideration of the elastic state of stress will generally be sufficient, but in coal mines, where hanging wall and footwall consist of sandstone, shale, or other sedimentary rocks of moderate strength, a considerable part of the mining operations will also be in the plastic zone. The sudden change in behavior of the rocks below a depth of 1500 to 1800 ft is a well-known fact in coal mines, which is also evident from the considerable difference in the maintenance costs of tunnels, drifts, and other mine openings above and below this "critical depth." Tunnels above the critical depth, which require only minor repairs as long as the surrounding rock is not disturbed by mining operations, are very difficult to keep open below the critical depth. Roof, floor, and sides begin to swell and heave, and the rock seems to "flow" towards the tunnel. The rock around the tunnel will eventually come to a rest, but the time required to re-establish the equilibrium is much longer than for a tunnel in the elastic zone.

The mechanism of flow and the re-establishment of the equilibrium in the plastic zone is governed by the laws of plasticity.

In order to arrive at the basic factors that govern the plastic behavior of rocks, Fenner first analyzed the stresses in an ideally homogeneous plastic medium. The relation between the horizontal stress (σ_H) and the vertical stress (σ_V) depends upon the coefficient of friction (μ) of the medium and can be expressed by the following formula:[10]

$$\frac{\sigma_V}{\sigma_H} = (\mu + \sqrt{1 + \mu^2})^2$$

To illustrate the behavior of a plastic medium around an opening, we assume that a vertical cylindrical shaft is sunk in this medium. At some depth a ring is left open in the shaft lining so that the medium can flow freely into the shaft.

The zone of particles engaged in this movement will thus constitute a circle, which gradually expands as the pressure release progresses.

The extent of the flow zone depends upon the radius of the shaft and the pressure exerted by the shaft lining since the flow is caused by the difference between the radial

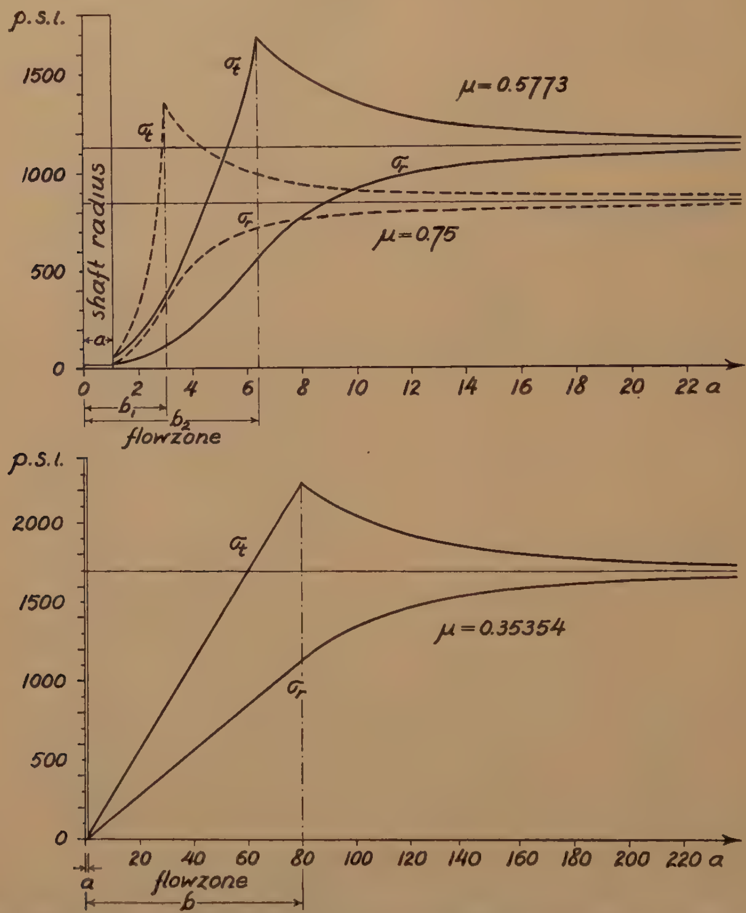

FIG 5—TANGENTIAL AND RADIAL STRESS ON AND BEYOND THE SHAFT LINING OF A VERTICAL SHAFT IN A PLASTIC MEDIUM (*After Fenner*[6]).

Since all particles move towards one point (the center of the shaft), the space available for each particle will decrease, or the tangential stress on each particle will increase. The zone of flow will thus continue to expand until the tangential stress on its boundary has obtained such a magnitude that no further increase of the flow zone is possible.

pressure at the wall of the shaft and the pressure in the undisturbed medium.

Fig 5 illustrates the extent of the flow zone[11] at a depth of 3300 ft for media with different friction coefficients. In this example the reaction of the shaft lining (σ_{R_0}) is assumed to be 14.2 psi (1 kg per cm^2), and the specific gravity of the medium is taken as 2.4, so that the vertical pressure

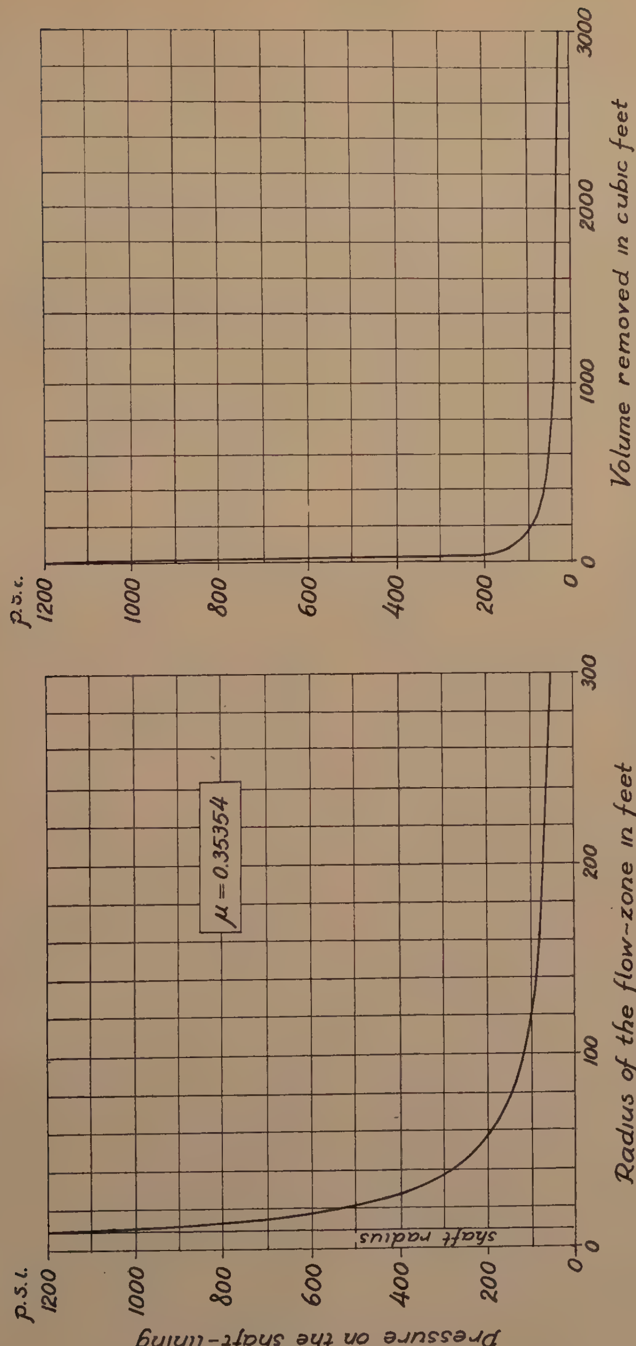

FIG 6—RELATION BETWEEN RADIUS OF FLOW ZONE AND VOLUME OF MATERIAL REMOVED (*After Fenner*[6]).

at 3300 ft is equal to 3400 psi. The radial stress (σ_r) and the tangential stress (σ_t) increase, extending outwards from the shaft, until the sum of these stresses becomes equal to the sum of the principal stresses ($2 \times \sigma_H$) in the undisturbed medium.

The radius of the flow zone, that is, the pressure release at the wall of the shaft, is directly related to the volume of material that is allowed to flow in the shaft.

The curves in Fig 6 show that the original pressure of 1700 psi on the lining of the shaft (radius: 10 ft, depth: 3300 ft, μ = 0.35354) can be reduced to 100 psi, if 200 cu ft of material is allowed to flow in for every 3.28 ft (1 m) of shaft. The original volume removed in sinking the shaft 3.28 ft is 1000 cu ft.

The analysis of the plastic flow is more complicated for a horizontal cylindrical tunnel because the two principal stresses in the plane of flow are not equal, as in the vertical shaft.

The results of the calculations are illustrated in Fig 7.[12] The relation between the reaction of the tunnel lining (σ_{R_a}) and the limit of the flow zone is plotted for a number of values of σ_{R_a}. The sum of the radial and tangential stress in the flow zone increases, extending outwards from the tunnel, until at the limit of the flow zone this sum has become equal to the sum of the principal stresses σ_H and σ_V at that depth.

The condition $\sigma_R + \sigma_T = \sigma_H + \sigma_V$ at the boundary of the flow zone is very important since it determines the minimum value of σ_{R_a} at which equilibrium can be restored. Calculations[13] show that, under the conditions assumed in the example, σ_{R_a} becomes less than 60 psi, the sum of the tangential and radial stress never becomes equal to $\sigma_H + \sigma_V$. The flow zone will thus reach the surface, and equilibrium will never be restored.

The effects of plastic flow encountered in the field will be different from the effects in the foregoing theoretical discussion, because rocks have appreciable elastic properties, which in the theoretical examples were assumed to be negligible. The pressure release caused by flow will eventually decrease the stresses to the limit where the rock in the vicinity of the opening returns to the elastic state. The pressure problem, as far as the opening is concerned, will thus be the same as in the elastic zone, and a pressure ring will be formed around the opening. At this final stage the opening will be surrounded by the following zones: first the zone of (elastic) pressure release within the pressure ring, outside this ring a zone in which the rock exists in the elastic state, around this zone the flow zone, and finally beyond the limit of the flow zone the undisturbed plastic zone.

Van Iterson's Theory—The theory advanced by Van Iterson consists of two parts. The first part[7] deals with the stress distribution around openings in an elastic medium. The results of his analysis are in principle the same as the results obtained by Fenner, and will therefore not be discussed.

The second part[8] deals with the subsidence phenomenon observed in longwall mining.

The subsidence theory given by Rice[14] 25 years ago is still widely accepted since no other theory has been advanced up to the present time. In the latest edition of Peele's Mining Engineers' Handbook[15] the mechanism of longwall subsidence is still explained according to Rice's views.

The mathematical explanation given by Van Iterson originated from the pseudoplastic behavior of rocks observed in mining operations. Fig 8 shows the hanging wall of an entry in sedimentary rock. The measures in the hanging wall, which were horizontal when the entry was made, "flowed" into the zone of pressure release. This could never have occurred if the rock had behaved as an elastic mass. Similar behavior occurs in the hanging wall and footwall at the coal front in longwall mining, closely

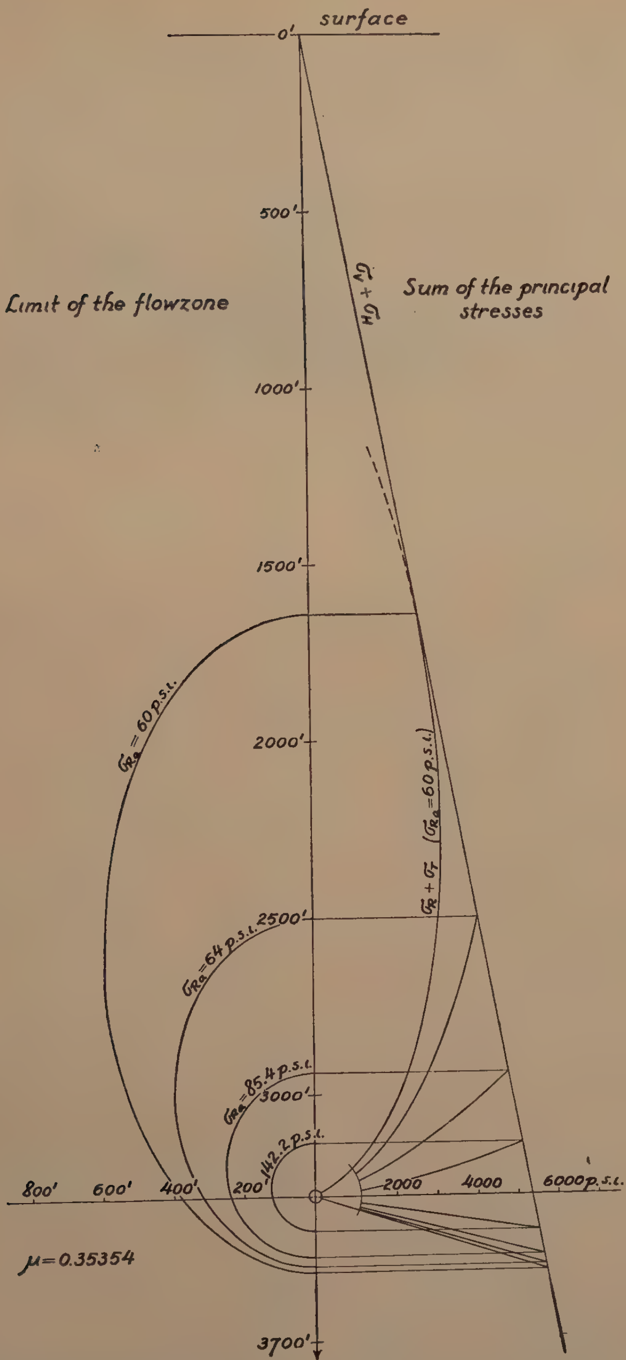

FIG 7—FLOW ZONE AROUND HORIZONTAL CYLINDRICAL TUNNEL IN PLASTIC MEDIUM (*After Fenner*[6]).

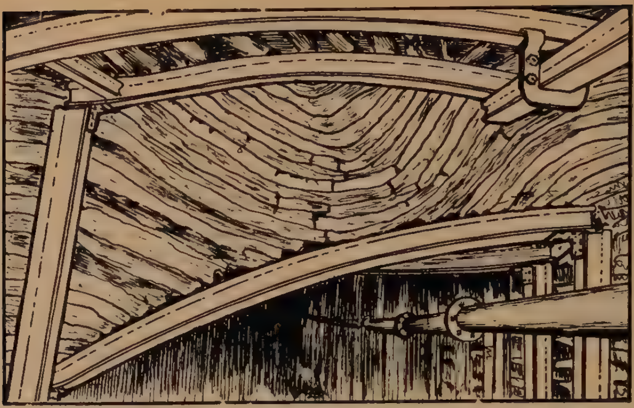

FIG 8—PSEUDO-PLASTIC BEHAVIOR OF ROCK IN ZONE OF PRESSURE RELEASE AROUND A DRIFT (*After Van Iterson*[8]).

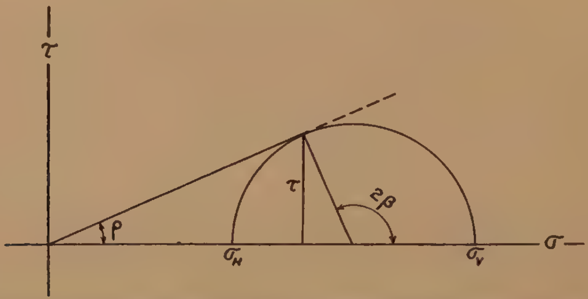

FIG 9—CIRCLE OF MOHR REPRESENTING STATE OF STRESS EQUILIBRIUM IN GRANULAR MASS WITHOUT COHESION AT MOMENT OF SLIDING.

associated with the typical longwall subsidence. The explanation given by Van Iterson is that the rock behaves as a granular mass rather than as elastic material.

The state of stress of a granular mass without cohesion at the moment of sliding is represented in Fig 9, in which ρ is the angle of natural repose of the material and β the angle the plane of sliding will make with the horizontal. Fig 9 (Mohr's circle) shows that $\beta = \frac{\pi}{4} + \frac{\rho}{2}$, or in other words: the plane of sliding (subsidence) will make an angle with the horizontal equal to 45° plus half the angle of natural repose.

A comparison between the theoretical angle of subsidence and the angle obtained from field measurements shows that Van Iterson's assumptions were correct:

Material	Angle of Natural Repose, Deg.	Observed Angle of Subsidence, Deg.
Sand	32	61
Coal	30–45	60–67½
Crushed limestone	30–45	60–67½
Crushed shale	38	64

The application of the principle to the pressure phenomena around small openings gives the same results as the principles outlined by Fenner for the plastic zone.

Summary of the Mathematical Stress Analysis—The results of the mathematical stress analysis may be summarized in the following statements:

1. The pressure dome or ring is in reality an ellipse with the major axis parallel to the direction of maximum stress. The cause of

failure of the rock within the dome lies in excessive tensile stress.

2. The dimensions of the pressure dome are independent of the depth below the surface.

3. The re-establishment of equilibrium in plastic rocks is created by flow. The pressure on the supports of the opening depends upon the flow that has been allowed to take place.

4. In rocks with appreciable elastic properties, the pressure release caused by flow will return the rock in the vicinity of the opening to the elastic state.

Conclusions

The theoretical analysis of the rock pressure problem will necessarily be confined to hypothetical ideal rocks under ideal conditions.

The behavior of rocks encountered in the field cannot be expressed in exact mathematical terms because of the many unknown factors. The knowledge of the principles of rock pressure has, however, little value if we do not know how to apply this knowledge to practical problems. Research should therefore be concentrated on the relation between theory and practice.

A careful investigation of all available field data, and an interpretation of these data based on the mathematical stress analysis, will undoubtedly enable us to formulate a set of practical rules for stress-control. This not only will lead to a reduction of operating costs by improving the methods of construction and support of mine openings, but will also give us valuable information for improving the most important operation in mining: the recovery of the ore.

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DISCUSSION

H. E. McKinstry*—Mr. Schoemaker is to be congratulated not only for winning the award of the National Student Prize Paper contest but for presenting an excellent critical review that brings the status of rock pressure up to date. As I recently published conclusions[19] virtually identical with the "new concept" so far as stress in strong rocks is concerned, I must take this opportunity to explain that my failure to credit the ideas either to Fenner or Van Iterson was due, embarrassingly enough, to my ignorance of the existence of their papers. No doubt many others interested in rock failure will, like myself, be grateful to Mr. Schoemaker for bringing the subject matter of

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these European contributions to the attention of American readers.

While the "new concept" is a definite advance in bringing ideas of rock failure into accord with the principles of elasticity and plasticity, it is, of necessity, based on certain assumptions which should be clearly recognized as assumptions and not mistaken for immutable laws of physics.

The first assumption, homogeneous rock, has been clearly specified in Schoemaker's paper. Naturally, any mathematical deductions need modification in the presence of such features as faults, joints, bedding planes, and schistosity.

A second and less obvious assumption concerns what might be called "environmental stress," the state of stress that exists in rock before any mine openings have penetrated it. Environmental stress is important because it is the starting point for any calculation of stress concentrations around openings. Much of the discrepancy between various theories of rock pressure results from failure to distinguish between environmental stress and locally concentrated stress. All theories assume, probably correctly, that the vertical component of environmental stress is equal to the weight of the overlying column of rock. But in regard to the horizontal component of stress, Fenner, in common with certain other authorities,[20,21] assumes that the adjoining rock offers a reaction just sufficient to prevent the expansion resulting from the vertical load, that is,

$$S_2 = S_3 = S_1 \frac{\nu}{1 - \nu} \qquad [1]$$

where S_1 is vertical component of stress, S_2 and S_3 are horizontal components and ν is Poisson's ratio (used in the American sense of the term).

Whether or not this assumption is correct cannot be determined readily by direct observation, for in order to gain access to the point where a measurement could be made it is necessary to make a hole which radically changes the stress distribution.

Eq 1 above would be realistic if the element in question (say an imaginary cube of rock at depth) had been created as an elastic body and then loaded by piling an elastic mass on top of it. To see why it may not apply even remotely in typical natural cases, let us consider a cubic foot within an intrusive granite body (assuming with the "pontificators" that there are such things as intrusive granites). When the granite was emplaced as a liquid magma, any cubic foot within it must have been under hydrostatic pressure. After it solidified, it was still under hydrostatic pressure, if we may neglect, for the moment, any stresses set up by cooling-contraction and crystallization. Since granite normally freezes at considerable depth let us assume arbitrarily that our cubic foot is at a depth of 20,000 ft. Erosion then begins removing cover and continues until the imaginary cube is within depth reachable by mining, say 5000 ft. Relief of vertical pressure will have allowed our cube to expand vertically, but the enclosing rock will prevent it from expanding horizontally. Assuming no change in horizontal dimension, and expressing the ratio of original depth to present depth by a factor C, the state of stress would be given by the equation:

$$S_2 = S_1 \frac{C(1 - 2\nu) + \nu}{(1 - \nu)} \qquad [2]$$

Under the depth conditions postulated, C will equal 4 and if Poisson's ratio (ν) be taken as 0.2, the horizontal component of stress will be 3.25 times the vertical. (It will be noted that if the original depth had been taken as zero instead of 20,000 ft, C would have been zero, Eq 2 would be identical with Eq 1, and if $\nu = 0.2$, horizontal stress would have been only one quarter of the vertical.)

Near the surface, vertical stress (S_1) becomes very small, but the factor C becomes enormous. The effect of horizontal stress is sufficient to produce horizontal sheet jointing, to induce rockbursts in quarries[22] and to crush septa of granite between vertical drill holes.

If instead of being granite or other igneous rock, the rock is metamorphic, the stress condition would not be greatly different, since metamorphic rocks presumably have accommodated themselves to a hydrostatic environment by recrystallization while at great depth.

It may be that the conditions represented by Eq 2 are extreme, since contraction-cooling would exert an opposing influence to the tendency toward expansion by unloading. Furthermore, it is possible that during a long sojourn at depths of only a few thousand feet, differential stress is relieved by intergranular adjustment, by slipping on minor joints and by

permanent "set," thus reestablishing hydrostatic pressure. But it is difficult to see how the horizontal stress can become much *less* than the vertical. In brief we do not know what to take as the "initial condition" from which to calculate environmental stress; stated otherwise, we do not know what value to use for C in Eq 2. Presumably C should be taken as positive and thus the stress condition lies somewhere between those of Eq 1 and Eq 2.

These sources of uncertainty may perhaps be classed as "inherent stress," the possible presence of which Schoemaker clearly recognizes, but the geologic history of most rock masses offers little support for the assumption that C is zero, that is, for setting up Eq 1 as the "ideal" condition. Eq 1, impressive as it appears, is a doubtful improvement over the older horseback assumption of hydrostatic pressure, and either of these alternative assumptions, if postulated, can be considered only as illustrative and is not to be taken too seriously.

Since there is a glaring gap in our knowledge of the fundamental basis for any theoretical calculations, further factual information is extremely desirable. Two lines of investigation suggest themselves:

1. Dimensional measurements by strain gauges on blocks of rock in variously oriented workings both before and after removing the blocks from their natural position. This should give a measure of locally concentrated strain from which the environmental stress can be calculated by taking into account the shape of the opening. (Measurements by this method have been undertaken, for example, in connection with tunnel work in Colorado.)

2. Observation of the rate of slabbing-failure in workings of alternative cross-sectional shape, a method suggested by Dr. K. K. Welker in personal conversation. Theoretically, a drift of circular or elliptical cross section will slab equally from all parts of its periphery when the relation between the axial ratio of the ellipse and environmental pressure is such that:

$$\frac{\text{horizontal axis}}{\text{vertical axis}} = \frac{\text{horizontal pressure}}{\text{vertical pressure}}$$

It is not sufficient, however, to assume that an opening will voluntarily assume its most stable shape. For example, under hydrostatic environmental stress an ellipse with its major axis vertical would experience greatest tangential stress at its top and bottom and failure there would tend to increase ellipticity rather than inducing the more stable circular form.

Still another assumption concerns the critical value for failure of strong rocks. The following remarks apply primarily to conditions in which environmental stress does not exceed rock strength and particularly to failure at or near the walls of openings.

Schoemaker correctly states that a plutonic rock having a shear strength of 2800 psi would be in a state of elastic stress down to a depth of 6500 ft under the conditions of Eq 1. Note, however, that this statement applies to environmental stress; once an opening is made, the local stress at its periphery will be very different. Schoemaker's diagram, Fig 4*a*, shows that the compressive stress at the sides of a circular opening would be 2.75 times the vertical pressure. It follows that a shearing stress of 2800 psi would be reached at a depth of 1750 ft. (If the environmental pressure be taken as hydrostatic, the corresponding depth would be 2400 ft.) True, this stress concentration does not extend far into the walls, but failure at the periphery would throw stress on the newly exposed surface. The increasing ellipticity would increase the stress concentration at the sides of the opening and failure would propagate itself. That drifts in strong rock do not in fact fail by shear at any such shallow depth seems due chiefly to the fact that shear strength, as determined by standard shearing tests, is not a full measure of the resistance of rock to shear in underground conditions, for in compression the normal stress on potential planes of failure greatly increases the shearing resistance. On the other hand, the standard laboratory measurements of failure in compression, which at their face value would indicate a shearing strength of roughly five times that in pure shear, probably give a higher value than would apply to mine openings because under the conditions of standard tests the end-plates of the machine impose a frictional resistance to any expansion normal to the axis of pressure. When this constraint is reduced by lubricating the bearing surfaces, or by using appropriate bedments, the apparent strength is materially diminished.[23]

Now it is a fact of significance for any theory

of rockbursts that where free lateral expansion is permitted, the initial failure of test pieces does not take place along surfaces of shear but on surfaces parallel to the direction of applied compression. Even where lateral expansion is retarded by lateral pressure, that is, where all principal stresses are compressional, experiments by Bridgman[24] have shown that brittle materials can still fail along tension or (as Bridgman prefers to call them) extension fractures.

Quite in accordance with this behavior is the common observation that the initial failure of pillars and the spalling of the walls of drifts take place along fractures roughly parallel to the periphery of the opening; this direction is parallel to the axis of principal stress, which, at the periphery, is necessarily tangential, since the stress across the free boundary is zero. The fragments that detach themselves, often with explosive violence, are thin and slab-like or shell-like—in no sense the wedge-like shapes that would result from failure by shear. Most authorities on rock failure have either ignored this mode of failure or endeavored to explain it away, perhaps because the textbooks on strength of materials say that unless actual tensional stress is operative, materials ought to fail by shear. It is true that pillars may fail ultimately by shear, but usually only after thinning and weakening by extensional failure.

Schoemaker shows correctly that under the conditions of Eq 1 actual tensile stress is present at the top (and bottom) of a circular opening. Such stress should produce vertical or radial tension fractures and it is true that such fractures are occasionally noted, but in deep mines slabbing of the roof is much more commonly developed along flat cracks parallel with the surface of the back. What does this mean? Since rocks are markedly weak in tension, it must mean either that drifts depart far enough from circular toward elliptical cross section to eliminate tensional stress (not a universal condition) or else that the conditions of Eq 1 are not common. This suggests that: (1) the horizontal component of environmental stress is normally greater than required by Eq 1, and (2) a "pressure dome" (in the sense of a region, not necessarily domelike, within which stress exceeds rock strength) is not exclusively a region of tensile stress.

Since initial failure in deep mines, especially in the case of explosive rockbursts, takes place so largely by extensional fracture, it would seem that the critical strength-characteristic is neither the strength in simple tension nor in pure shear nor in compression-crushing, but what might be called, perhaps paradoxically, "tensile strength in compression." Since extensional failure is not fully explained by conventional theories of rupture, it calls for further experimental and mathematical investigation.

This discussion is not intended to be a criticism of Schoemaker's interesting and instructive article, but rather to point out one or two blind spots in all of the common theories of rock bursts.

R. P. SCHOEMAKER (*author's reply*)—Mr. McKinstry has raised several important arguments against the "new concept" of rock pressure. The gap between Fenner's theory (and other similar theories) and the phenomena known from field observations is evident, although not necessarily indicative of major shortcomings in the theory. On the other hand, if the basic assumptions should be incorrect, as suggested by McKinstry, the whole theory would fall down.

Under the conditions postulated in McKinstry's illustrative example the ratio between horizontal and vertical stress would indeed become enormous. However, in this analysis the possible effects of plastic, or pseudo-plastic, behavior has not been considered, other than in stress readjustment *after* the imaginary intrusive has come close to the surface.

When temperature and pressure decrease, the intrusive in the liquid state (under hydrostatic pressure) will not suddenly become an elastic mass, but will go first through a plastic phase and possibly through a phase in which the rock behaves as a granular mass without cohesion. The last approach has been used by Van Iterson in dealing with pseudo-plastic behavior of sedimentary rocks.

The relation between the vertical stress (S_V) and the horizontal stress (S_H) in the plastic phase is governed by the internal friction (μ), for which Fenner gives the equation:

$$\frac{S_V}{S_H} = (\mu + \sqrt{1 + \mu^2})^2$$

The magnitude of μ for plutonic rock is not

known, but we will assume it to be equal to μ for sedimentary rock of medium strength (0.75). In the liquid state μ will be zero, or $S_H = S_V$, and with increasing plasticity μ will increase to a maximum of 0.75, under which conditions the ratio will become: $S_H = \frac{S_H}{4}$.

The question is: how does the plastic phase affect the horizontal stress in the rising intrusive? If the plastic mass behaves according to Fenner's views, considerable relief of horizontal pressure will be possible, so that at the transition from the plastic phase to the elastic phase the relation:

$$S_2 = S_3 = S_1 \frac{\nu}{1 - \nu}$$

(Eq 1 in McKinstry's discussion) still holds true.

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Determination of Room and Pillar Dimensions for the Oil-shale Mine at Rifle, Colorado

By Fred D. Wright,* Junior Member and Philip B. Bucky,† Member AIME

(New York Meeting, February 1948)

Foreword

By E. D. Gardner‡

The present known petroleum reserves are limited, and unless important new fields are discovered the Nation will be dependent, in the not too distant future, upon imports or upon synthetic liquid fuels to supplement domestic petroleum supplies. As a national defense measure, it would appear desirable to develop a substitute source of liquid fuels within the borders of the country if new discoveries of petroleum should prove disappointing. Congress undoubtedly had this in mind when it passed the Synthetic Liquid Fuels Act (78th Congress, approved April 5, 1944). The act directed the Bureau of Mines to construct and operate demonstration plants for producing synthetic liquid fuels from coal and oil shale, and from forestry and agricultural products. As a part of this program, the Bureau of Mines has built a demonstration plant at Anvil Points, near Rifle, Colo., for retorting the oil shale, and has developed a mine to supply the plant with oil shale.

At the beginning of mining operations, the physical qualities of the oil shale, which would have a bearing upon the selection of mining methods and practices for exploiting the deposits, were largely unknown. A comprehensive mining-research program has been set up to determine the best methods and practices to follow to obtain the lowest overall practicable mining cost for producing the oil shale on a large scale.

Unusually low mining costs will be necessary for shale oil to be produced commercially. Consideration now is being given to exploiting a series of flat-lying beds 70 ft thick by a room-and-pillar method of three benches with the advance heading at the top.

This report is one of a series giving the results obtained in research on mining problems.

Introduction

The structure and physical qualities of the oil shale indicate that the 70-ft interval of the oil-shale measure can be safely mined in open stopes; studies indicate that this method would be the most economical one to use.[1] As labor will be a major item of expense, the largest equipment that could be operated efficiently in the workings would be required to obtain the lowest practicable mining cost. It was therefore desirable to know the maximum safe span of unsupported rooms and the size of pillars necessary to support the overburden. This paper describes the work done at the Columbia School of Mines Barodynamic Laboratory in determining the maximum safe unsupported roof spans and the minimum safe pillar dimensions for the oil-shale mine at Rifle, Colo.

Published by permission of the Director of the Bureau of Mines, U. S. Department of the Interior. Manuscript received at the office of the Institute Feb. 18, 1948. Issued as TP 2489 in Mining Technology, November 1948.

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[1] References are at the end of the paper.

Structure and Sampling of Deposit

Structure

The oil shale near Rifle, Colo., is part of a thick and extensive formation which has been described in detail by others.[2,3,4,5] The 70-ft thickness of shale to be mined outcrops near the bottom of a long line of cliffs and is overlain by 300 to 1000 ft of overburden. The formation is almost horizontal, dipping 3° at the mine area.

Inspection of the cores of three holes at the mine area and of a raise into the roof rock revealed that the 70-ft thickness of minable oil shale is overlain by a solid bed of shale which varies in thickness from 6.8 to 8.5 ft. This immediate roof bed is overlain by two thinner beds of variable thickness, which are in turn overlain by many thick beds of oil shale.

There was considerable fragmentation of the drill cores to a depth of about 100 ft below the surface. This condition apparently is due to weathering rather than to an inherent difference in the shale beds, as the line of demarcation between the weathered and unweathered zone has no correlation to a stratigraphic horizon.

Series of approximately parallel jointing planes occur in the mine workings. The interval between joints varies considerably; the average is over 5 ft. The dip of the joints is from 75 to 90°. Their strike varies in different parts of the mining area. Most of the jointing planes are localized and are not continuous throughout the total 70-ft thickness that is to be mined. Occasional water seepages indicate that a few of the joints may extend to the surface.

Vugs were seen along a stratigraphic zone 35 to 45 ft below the roof stone. These vugs are from 2 in. to 6 ft in diameter. Their cumulative horizontal length along one rib of an adit amounts to almost 30 pct of the rib length and on the other side to about 12 pct of the rib length.

Sampling the Deposit

Samples for testing of both the minable oil shale and the roof rock were taken, and their locations in the stratigraphic column were noted. Most of the samples came from two large adits and from a raise into the roof stone.

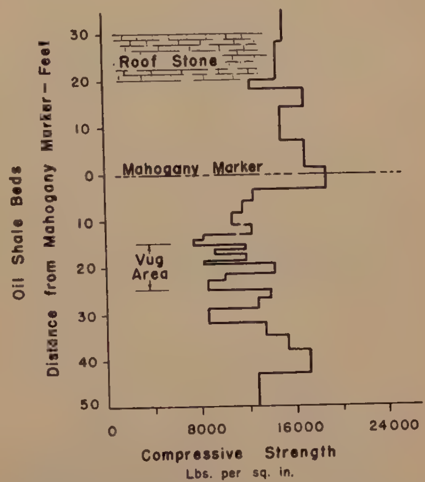

Fig 1—Graph showing compressive strength of pillar and roof stone along vertical section.

Physical Characteristics of the Oil Shale

Certain physical characteristics of the mine rock were determined as a preliminary to making the tests on small models of oil shale upon which the safe room width and pillar design were based.

Compressive Strength

A hydraulic press was used for compression tests. Test specimens were square prisms 1 by 1 by 2 in. The compressive strengths of the minable oil shale and immediate roof beds vary from 7350 to 19,000 psi and are presented diagrammatically in Fig 1.

Shear Strength

For determining shear strength, flat specimens 1 by ½ in. in cross-sectional area were loaded in double shear. Table 1 gives the results of some of the shear tests, which

varied from 890 to 4640 psi. It will be noted that the shale is very much stronger in shear perpendicular to the bedding than parallel to the bedding.

TABLE 1—*Results of Shear Tests on Typical Oil-shale and Roof-rock Samples*

Description of Sample	Direction of Shearing Force Relative to the Bedding Planes	Shear Strength	Number of Samples Tested	Standard Deviation, Pct
Roof....	Perpendicular	4,640	5	3.1
Stone...	Parallel	1,770	5	10.5
Oil Shale	Perpendicular	3,145	5	6.0
	Parallel	890	5	8.3
Oil Shale	Perpendicular	3,205	5	5.2
	Parallel	920	5	8.1

TABLE 2—*Values of the Modulus of Rupture of Different Size Beams Cut from a Sample of the Roof Rock*

Ratio of Span to Depth of Test Beam	Modulus of Rupture Pounds per Square Inch
13.3	6,590
19.7	4,730
24.6	4,320
34.8	4,240
40.0	4,070
50.1	3,370
52.6	3,350

Modulus of Rupture

Long, flat, rectangular test specimens, 1 in. wide, 8 in. long, and of varying depths, were used in making modulus-of-rupture tests. Specimens were supported at each end by a half-round steel rocker resting on knife edges running down the center of a base plate; they were loaded at the center of the span.

Several series of modulus-of-rupture tests were run but the values varied considerably for any given sample. The tests indicated that the modulus of rupture increased as the ratio of span to depth of beam decreased. The values in Table 2 are characteristic of the results obtained.

Modulus of Elasticity and Poisson's Ratio

The modulus of elasticity and Poisson's ratio of the samples were determined by loading prism-shaped specimens in compression and measuring strains. Both SR-4 resistance wire gauges and Porter-Lipp gauges were used for measuring the strains. The roof stone and medium-grade oil shale behaved elastically to stresses beyond 5000 psi. The rich oil shale, on the other hand, showed evidence of plastic deformation and permanent set at a stress of 3000 psi.

The modulus of elasticity of the roof rock and the oil shale varied from 500,000 to 3,500,000 psi. Poisson's ratio varied from 0.20 to 0.40. In general, the modulus of elasticity decreased in value, and Poisson's ratio increased with an increase in the assay value of the shale.

Other Physical Characteristics

Moisture appeared to have little effect on either the roof rock or the oil shale. Samples of both were immersed in water for over two months and tested in bending. No appreciable difference in strength was noted between these and freshly cut specimens, nor was there any visible evidence of a change in the physical aspect of the rocks.

Air-drying, however, caused checking in some of the richer samples of the oil shale. This was especially noticeable on the smooth faces of specimens which had been ground by an emery wheel. The shale tended to open up along its laminations and also, to a less extent, to crack across the bedding. The long, thin, rectangular specimens of oil shale used in the modulus of rupture tests warped badly if left exposed overnight and often cracked along the laminations. On the other hand, except for slight warping of thin specimens, samples of the roof stone showed no appreciable evidence of being affected by air-drying. Test specimens cut and air-dried for six months or more had the same compressive and bending strength as freshly cut samples.

ROOF AND PILLAR DESIGN

Centrifugal Method of Testing

It has been shown previously[6,7] that a scalar model built of the same material as

the structure being investigated—that is, the prototype—will behave in a manner similar to the prototype, if the density or effective weight of the model is increased in the same proportion as its linear dimensions are decreased. This is accomplished by substituting a centrifugal for a gravitational field, by placing the model in a centrifuge and controlling the strength of the centrifugal field by varying the rate of rotation. The number of times the effective weight of the model is increased is called the Model Ratio. The model dimensions times the Model Ratio determines the dimensions of the prototype that will behave similarly to the model being tested.

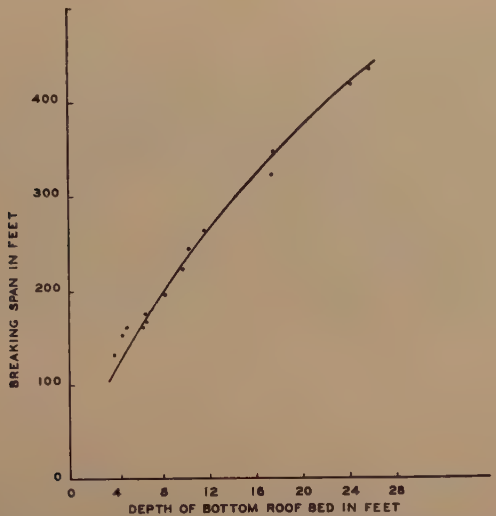

FIG 2—CURVE SHOWING BREAKING SPAN OF MINE ROOF AS DETERMINED BY THICKNESS OF BOTTOM BED OF IMMEDIATE ROOF.

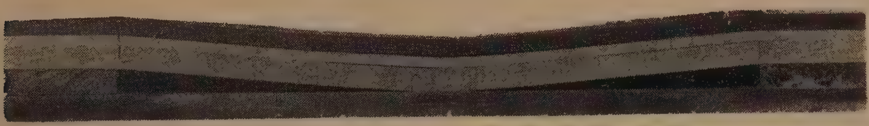

FIG 3—MODEL OF THREE-BED IMMEDIATE ROOF AFTER TESTING IN CENTRIFUGE. FAILURE HAS OCCURRED BY COLLAPSE OF ALL THREE BEDS.

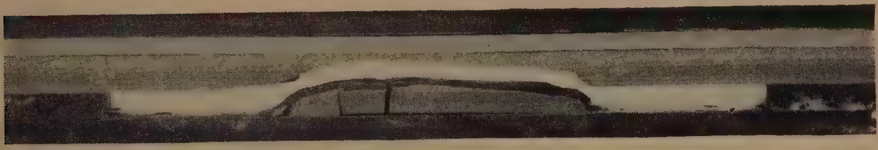

FIG 4—MODEL OF THREE-BED IMMEDIATE ROOF WHICH FAILED BY NATURAL ARCHING OF BOTTOM BED.

Determination of the Allowable Roof Span

The beds over a room being excavated from a flat-lying deposit can be considered as beams lying one above the other.[8] The fact that the beds are actually flat plates introduces an error into the concept which, however, is usually small and is always on the safe side.

In a room where no artificial supports are used, the immediate or effective roof includes only the first bed over the room and those beds which load the first bed. From the examination of the structure at the oil-shale mine, it was determined that the immediate roof consisted of a bed 6.8 to 8.5 ft in thickness, probably loaded by two thinner beds, which varied from a few inches to 6 ft in thickness.

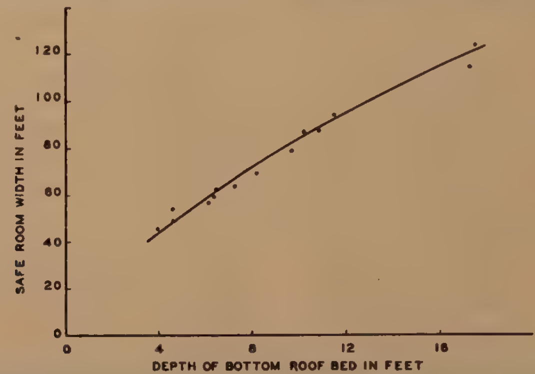

FIG 5—CURVE SHOWING MAXIMUM SAFE ROOM WIDTHS AVAILABLE FOR MINING AS DETERMINED BY THICKNESS OF BOTTOM ROOF BED. A SAFETY FACTOR OF 4 WAS USED.

Single-beam models cut to identical dimensions from each of the roof rock samples first were tested; these were run in the centrifuge to failure to compare their relative strengths.

For the next series of tests, models were made of a roof consisting of a bottom bed loaded by two weaker beds, corresponding to the respective mine strata; they were all run to failure in the centrifuge. Fig 2 shows the prototype breaking span for various thicknesses of the bottom bed. In these tests, it was noted that all the models that had a prototype depth of less than 10 ft failed by the collapse of the three beds (Fig 3), while the models having a prototype depth of over 11 ft failed by natural arching of the bottom bed (Fig 4).

A safety factor of 4 was used in determining the maximum safe spans that should be used in the oil-shale mine to take care of factors not accounted for in the tests. Rectangular pillars laid out on a checkerboard system are to be used in the mining method to be tried first at the mine. In this case, the maximum unsupported roof span is considered to be the distance between the corners of diagonally opposite pillars. The maximum safe unsupported roof span is reduced accordingly in determining the safe room widths (Fig 5).

Safe Pillar Design

Structurally, the pillars of a room-and-pillar mine can have two principal functions. In a mine of small lateral dimension, with a geologic structure similar to the oil-shale deposit, the overburden is largely supported by the unexcavated rock along the mine boundaries, and the pillars act primarily to support the immediate roof. As a mine becomes larger in both lateral directions, the pillars are required to assume a greater share of the overburden load until the support provided to the mine overburden by the unmined rock along its boundaries becomes negligible, except to the pillars near the boundaries. In this investigation, it was assumed that the oil-shale mine will be extensive in both lateral directions and that the pillars will support the total weight of the overburden.

Previous experimentation on photoelastic models[9] has shown that, under certain con-

ditions, the ribs of a pillar are under a much higher stress than the center. Several models were therefore made to determine whether such was the case for conditions similar to those at the oil-shale mine.

load on the pillars by the allowable stress for the pillar.

Another group of model tests was therefore made to determine the influence of time on the pillar rock. The pillars were made

FIG 6—FOUR-PILLAR MODEL UPON COMPLETION OF FINAL TIME TEST.

Deformation of pillars is evident, as are V-shaped cracks in right center pillar. The black spots at bottom of pillars are glue which was used to tack pillars in place while assembling the model. No glue was used underneath pillars. The upper portion of overburden is obscured by metal plate which was part of model holder.

In these models the overburden was supported by several pillars of equal dimensions and overhung the outside pillars by one-half the room span. Thus the models represented part of a vertical cross-section through a room-and-pillar mine of large lateral extent.

In order to observe the initial evidences of failure in the pillars, the models were run for definite time intervals at speeds below the calculated speed for failure, removed from the centrifuge and inspected. The speeds were then progressively increased until the models failed. These tests gave no visible evidence that there was compressive stress at the ribs in excess to the compressive stress at the centers of the pillars. Some of the pillars failed by shearing diagonally across their breadth, while others failed similarly to the usual hourglass failure of a brittle material broken in a compression test.

A striking feature of the tests was the influence of the time factor on the strength of the pillars. Except for this factor, it appeared that the safe pillar area could be determined accurately by dividing the total

from the samples of the pillar rock having the lowest compressive strength.

The models approximated the conditions of the Rifle deposit by having several relatively thin beds over the pillars which, by their deflection, might cause excess loading at the pillar ribs (Fig 6).

The first model was run at a gradually increasing speed until the pillars failed. The calculated average compressive stress in the pillars at failure was within 2.4 pct of the ultimate compressive strength of the pillar as determined by compression tests in a hydraulic press. This value is within the percentage accuracy of the compression tests.

The other models were run at successively lower speeds. For each model, the centrifuge speed was kept constant and the time it took the pillars to fail was noted. Prototype time was determined by multiplying the time it took the pillars to fail by the model ratio. Fig 7 shows the relation between the prototype time of loading and the compressive stress at which the pillars failed, expressed as a percentage of the ultimate compressive strength of the pillar

rock. The slope of the curve indicates that a pillar stressed to 75 pct of its ultimate compressive strength would stand for a great many years.

beds as well as a zone in which large openings appear. As might be expected, horizontal planes of weakness have little, if any, effect on pillar strength. Test specimens broken along horizontal bedding planes had the same compressive strengths as unbroken specimens.

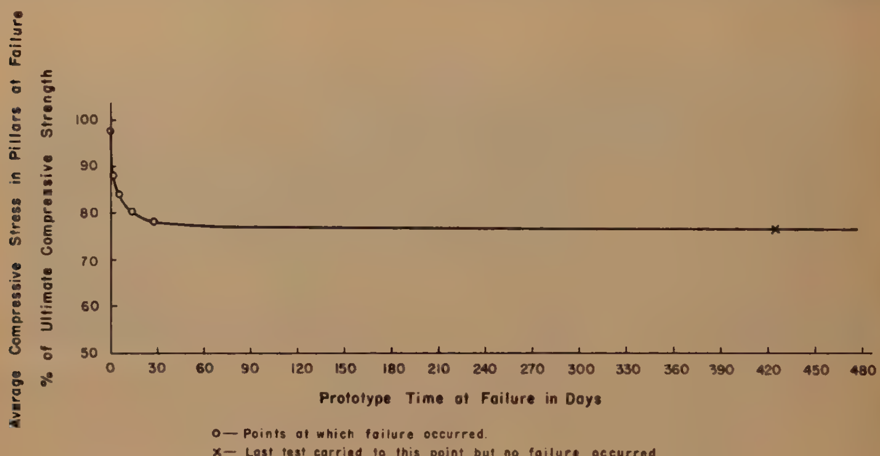

FIG 7—CURVE SHOWING EFFECT OF TIME ON PILLAR FAILURE.

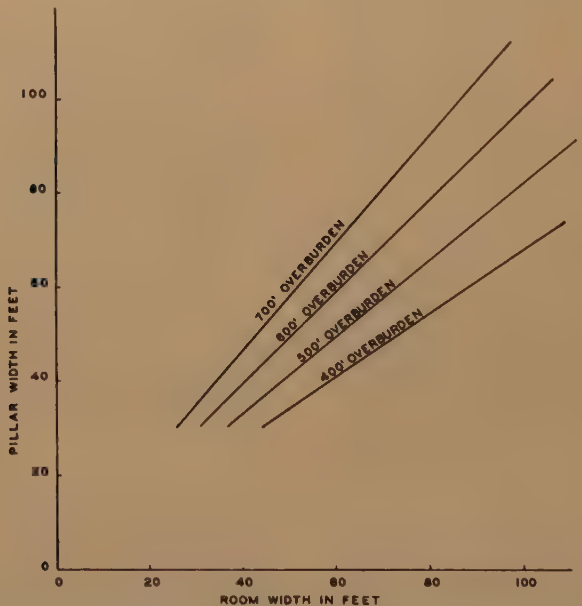

FIG 8—CURVES SHOWING MINIMUM SAFE PILLAR WIDTHS FOR SQUARE PILLARS AS DETERMINED BY ROOM WIDTH CHOSEN FOR OPERATION AND BY DEPTH OF OVERBURDEN.

In the discussion on the physical structure of the deposit, it was mentioned that there are horizontal and steeply dipping planes of weakness through the oil-shale

The occasional jointing planes that ap-

pear to cut through all the minable beds could weaken the pillars somewhat by splitting them, thereby increasing their height: width ratio and causing local spalling if close to a rib.

The vugs that appear in the shale are distributed vertically throughout a 10-ft zone, so their reduction of the effective pillar area is not so great as their cumulative area would indicate.

A safety factor of 3 was considered adequate for the pillars. This is less than that chosen for the roof rock because the factors that determine pillar strength are not so dependent on factors not included in the tests of models. The lowest value for the ultimate compressive strength of the oil shale determined by compression tests was 7350 psi. From Fig 1 it is seen that this was for a very thin bed and that there are only five beds of 3-ft thickness and under, whose compressive strengths are less than 10,000 psi. Experiments by Dr. Leonard Obert* of the Bureau of Mines, College Park, Md., have shown that the overall strength of a pillar is not materially affected by a few thin beds, even if they are considerably weaker than the average strength of the beds comprising the rest of the pillar.

To determine the safe allowable compressive stress in the pillar, the value of 10,000 psi was taken as the overall strength of the pillar rock. This value was reduced by 25 pct to account for the factor of time on failure. With a safety factor of 3, the allowable stress was therefore 2500 psi. In Fig 8, the safe square pillar widths are plotted versus room width for various heights of overburden. Knowing the depth of overburden and the width of the rooms to be mined, the minimum pillar dimensions thus can be determined readily for different sections of the mine.

Conclusion

Because of the effects of surface weathering the recommendations in this report are valid only if the workings are more than 200 ft from the surface.

The maximum safe unsupported roof span is determined by the immediate roof. Cores of the immediate roof should be obtained at 200-ft intervals to check the depth and the geological character of the immediate roof. Large rooms can be excavated safely as long as the roof is geologically sound and not damaged by blasting. Fig 5 then may be used to determine safe room widths.

The recommended square pillar widths are given in Fig 8. Jointing planes running parallel and close to pillar ribs may cause some spalling. The richer zones of shale should also be watched closely since air-drying will cause them to "check" at their exposed surfaces and possibly to spall.

* Personal communication.

Acknowledgments

The authors are indebted to Dr. E. C. Ingalls for the use of the Electrical Strain Gage Equipment at the Columbia University Civil Engineering Laboratory.

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Ground Movement Adjacent to a Caving Block in the Climax Molybdenum Mine

By John W. Vanderwilt,* Member A.I.M.E.

(Chicago Meeting, February 1946)

The unpredictable behavior of ground movement and subsidence has complicated the problems that attend the extraction of large quantities of ore. Special studies, particularly relating to coal mining, have yielded such a voluminous literature and so many involved controversial theories that a partial review would be difficult. Limited space makes any kind of a review impractical.

The features to be considered cover only a relatively small part of the broad problems of ground movement and subsidence, but it is believed that this small part is critical. The practical aspects are important, since they bear directly on safety of mine workings, on recovery of ore and on dilution of ore in block caving and related systems for extracting large quantities of rock.

The ground movement to be described occurs outside the vertical boundaries of a caved block in the area that is commonly referred to as being in the "zone of draw." "Draw" is a term that originated in the subsidence problems related to coal mining, and it is defined by a line drawn from the margin of the area caved underground to the most distant fracture at surface on the same side of the caved area as shown in Fig. 1.[1] A line thus drawn is called the cave line, and its angle with respect to the horizontal is the cave angle or angle of draw. It is understood that the line is taken normal to the limit of mining, and in a vertical plane.

The prevailing theories as to draw and angle of draw assume that appreciable movement occurred along the cave line, which has been described as a fracture resulting from the shearing action brought about by the weight of the overlying load. Fig. 2 shows diagrammatically the mechanics of ground movement adjacent to an area of subsidence according to the prevailing theory of draw (*A*, from Royce[2]) and, for comparison, ground movement observed in the Climax mine (*B*). The text in Peele's Handbook[2] referring to Fig. 2 does not state the meaning of the "original position of top of workable ore" in the rectangular space above "ore in place," and the arrows are not explained. However, this presentation is most concerned with the zones outside the rectangle. According to the prevailing theory, as shown at the extreme left under *A*, Fig. 2, the vertical tension cracks at surface change at a shallow depth to shear fractures forming the cave line, and it is assumed that appreciable movement takes place along the cave line toward and into the subsiding caved block.

The mechanics of ground movement observed in the Climax mine, as shown under *B*, Fig. 2, does not accord with the prevailing theory, particularly as regards the cave angle and draw. The vertical tension cracks seen at surface die out at depths of 150 to 225 ft. and shear fractures do not develop. Movement is confined at surface to a mantle of rock 50 to 100 ft.

Manuscript received at the office of the Institute Dec. 1, 1944. Issued as TP 2000 in Mining Technology, May 1946.

* Consulting Geologist, Denver, Colorado.

[1] References are at the end of the paper.

thick, broken by the action of blocks tipping and falling down the slope toward the drawn area.

the distances at which ground movement may occur at surface in mining beds of coal of various thicknesses and at various

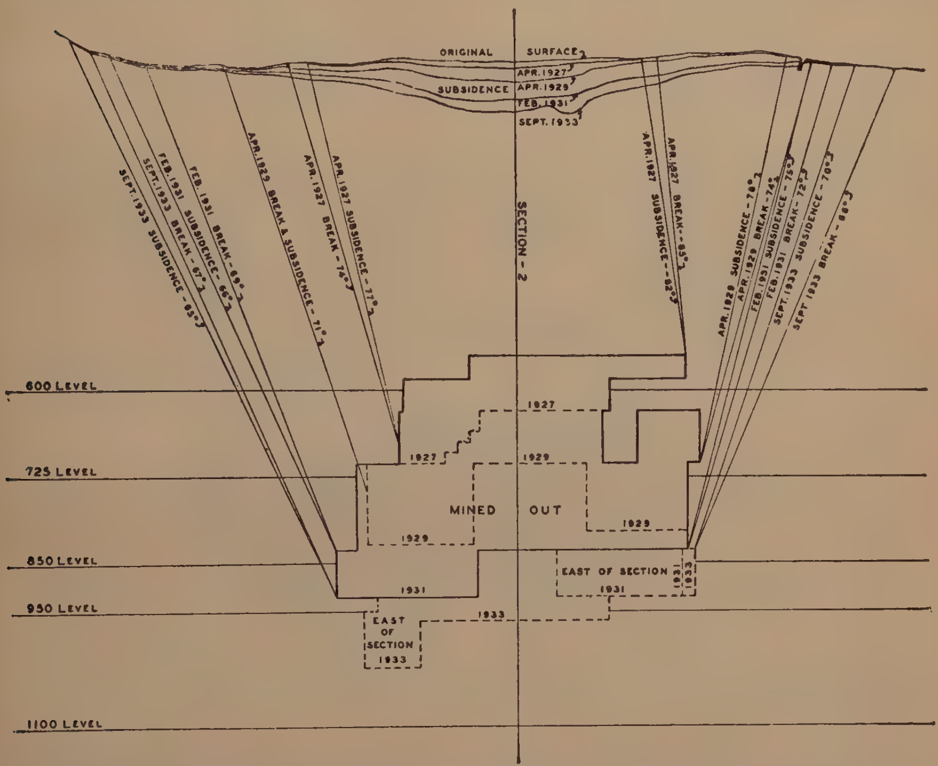

FIG. 1.—TYPICAL EXAMPLE OF SUBSIDENCE, CAVE LINES, AND ANGLES OF DRAW ACCORDING TO PREVAILING THEORIES OF GROUND MOVEMENT. (FROM KANTNER.[1])

REVIEW OF PREVAILING THEORIES OF DRAW ANGLE, WITH SUPPORTING EVIDENCE

The concept of draw and draw angle probably originated in connection with subsidence related to coal mining. Small vertical cracks commonly form at surface at varying distances outside the limits of coal mining, causing damage to buildings and other surface installations, which at times has led to costly lawsuits. As a result, the problem of draw in connection with coal mining has been studied intensively, and a number of different mathematical formulas have been invented to calculate depths. Attempts have been made also to include in the mathematical formulas the influence of dip and other structural features.

An excellent summary of earth and rock pressure and its bearing on angle of draw is presented by Moulton,[3] who believes that a common law governs the failure of all materials possessing cohesion. Collapse of walls of large opencuts in unconsolidated materials is given as evidence that this common law brings about an angle of draw close to 1½ to 1, or 60°. According to Moulton, the angle of draw in hard rock would be the same as the

angle of draw for moist sand and any other unconsolidated material, other than material possessing a semifluid nature.

In considering some problems in ground movement and subsidence, Rice,[4] in 1923, suggested a modification of the straight cave line. In its stead he visualized a curved surface of stability extending from the edge of the mine excavation and curving upward to the most distant vertical crack at surface. The conventional cave line is a chord to the curved boundaries suggested by Rice, as shown under *A* in Fig. 2.

In metal mining, including the iron mines of Michigan, cracks causing damage to shafts, buildings, and underground mine workings have occurred repeatedly at considerable distances from large stopes. From these experiences it is recognized that ground movement sufficient to cause damage to buildings and shafts can be expected well outside the caved area, at distances varying from 35° to 75° as expressed by conventional cave lines.

Accounts of shear fractures, either with or without displacement, to correspond with a cave line as defined by the prevailing theory of draw are conspicuous by their absence in recent literature. The absence of reference to specific examples of either movement along an inclined surface or of low-angle shear fractures without movement is in itself enough to raise the suspicion that the conventional theory of draw is not valid.

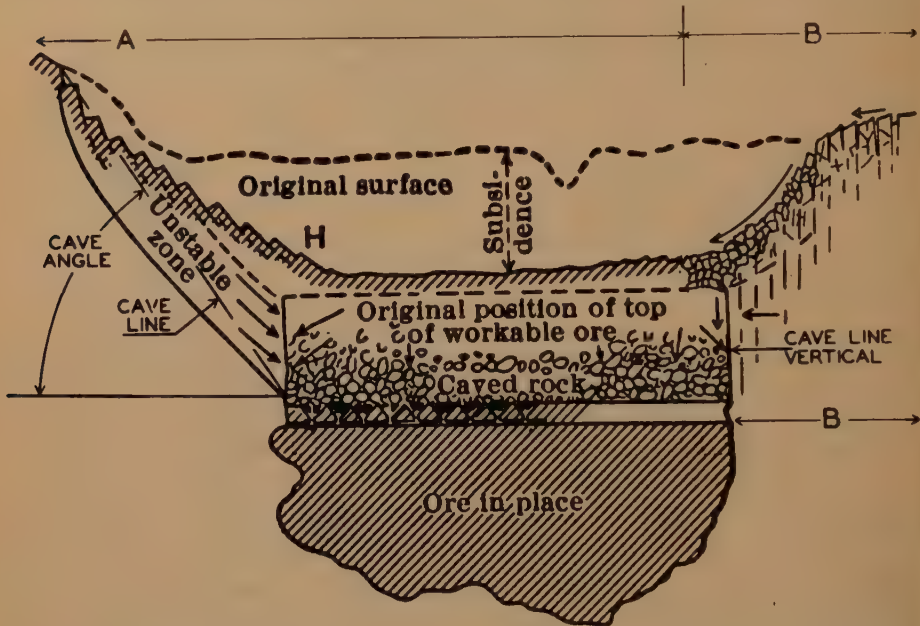

Fig. 2.—Subsidence in block-caving a large ore body according to: (*A*) generally accepted theories of draw (from Royce[2]), and (*B*) data observed in the Climax Molybdenum mine.

The collapse of steep walls of large open excavations in either unconsolidated material, as cited by Moulton, or in hard rock cannot be compared directly with ground movement adjacent to a subsiding area filled with broken rock. The side pressure of broken rock in the caved block would actively resist movement of any kind.

Review of Evidence Against Prevailing Theories of Draw Angle

A few of the numerous examples of essentially vertical caving that challenge the validity of the prevailing theory of draw will be cited. A real challenge deserving serious consideration comes from the iron mines in Michigan, in which the subsidence is along essentially vertical cave lines in the Athens mine at Negaunee, Mich.[5] In the Athens mine a stope 200 by 350 ft. and less than 200 ft. high caved vertically to surface through a jasper capping nearly 1900 ft. thick. Similar occurrences of subsidence along vertical lines are mentioned by Allen[5] in mines about one half mile north of the Athens mine, where the ore body lies at a depth of 800 to 900 ft. below surface.

Another example of vertical caving is recorded by Rice.[6] The top of the ore body in the Brier Hill mine, Norway, Mich., was about 800 ft. below surface and it followed down the 60° to 70° dip of the bedding underneath a hanging wall of slate. The main-line tracks of a railroad ran directly over the ore body. According to one theory of angle of draw, the cave would follow up the 60° to 70° dip and reach the surface 200 ft. or more from the railroad tracks, therefore the tracks would not be endangered. The management contended that caving might proceed vertically upward across the slate bedding. In order to protect the railroad, diamond-drill holes were used to check the progress of caving. In spite of waste fill used to prevent caving, vertical subsidence was detected in a few years in the diamond-drill holes. From 1916 to 1923, caving advanced progressively upward to the top of bedrock underneath the thick cover of glacial gravel, and the latest observations in 1930 indicated no additional advance in subsidence. An observation drift across the bed designed to observe caving upward along the bedding failed to detect any movement.

Herbert and Rutledge,[7] after detailed studies in four coal fields in Illinois, report finding draw in only one of the four areas, and the lowest draw angle reported in this one area is 81°.

Maclennan[8] describes an attempt to cause caving on a plane of 70°, but the draw was vertical in spite of the 70° cutoff. In this case the ore lying between the vertical and the 70° plane was not recovered by the cutoff, and, in addition, the money spent for the cutoff also was lost.

Subsidence in the Climax Mine

The Ore Body and Cave Area

Mining at Climax, since about 1931, has been through the Phillipson tunnel, which is at an elevation of 11,463 ft. The ore zone on the Phillipson level is an elliptical band 300 to 500 ft. wide, with an inside diameter of about 1200 by 1600 ft., and 800 by 1000 ft. on the White level, 400 ft. above. The area inside the ore zone is a fine-grained quartz formed by replacement of granite and schist, which forms the core around which the ore zone extends downward at dips varying from 60° to vertical.

The upper part of the ore zone and siliceous core are truncated by erosion, and the outcrops extend from Tenmile Valley up the adjoining side of Bartlet Mountain. The height of the top of the ore above the Phillipson level varies from about 350 ft. in the valley to about 850 ft. on the flank of the mountain.

Mining has advanced around the south, west, and northerly side of the elliptical ore zone, and, in round numbers, 45 million tons of ore have been extracted. The resultant cave that shows at surface is roughly the shape of a horseshoe open to the east. The lineal extent of this cave at surface is fully 6000 ft., with width 400 to 600 ft. The eastern, or

open, part of the horseshoe that remains to be caved is about 2800 ft. long.

The northern part of the cave lies on the slope of Bartlet Mountain, at an elevation well above 12,500 ft. The southern part of the cave, and the part described in this report, lies on the lower slopes and in the valley of Tenmile Creek, at elevations between 11,700 and 11,800 ft. The overburden or waste varies from a maximum of about 900 ft. to the north and a minimum of 10 to 40 ft. to the south.

The characteristics of the escarpments forming the walls of the cave vary. Vertical walls 50 to 100 ft. high have been observed. As a rule, rock-slump and rockslides falling around a subsiding area reduces the over-all slopes to an average of 40° to 60°.

Alteration and Fracturing of the Rock

The character of the rock is an essential factor in any consideration of ground movement and subsidence, therefore a brief summary of the physical makeup of the rock will be given.

The host rock at Climax is pre-Cambrian granite and schist, and a quartz porphyry of Tertiary age. In the caved block under consideration the quartz porphyry is confined to the lower 100 to 200 ft., and granite predominates above the porphyry.

Alteration increases gradationally from the hanging-wall side of the ore zone to the footwall side (Fig. 3, left to right) and the footwall is a massive fine-grained quartz replacement of the host rock. Alteration consisting of quartz and orthoclase replacement as well as a general decomposition of feldspars makes it difficult to distinguish rock types in places in the ore zone.

The rock throughout the ore zone is intersected by closely spaced fractures, which show no predominant pattern or orientation except in local areas. In the ore zone the spacing of the fractures on the average varies from less than an inch to about 6 in. The fine-grained quartz in the footwall has a fracture spacing of about 6 to 24 in., and this quartzose rock is stronger than the rock in the ore zone.

Everywhere the rock between fractures is moderately hard to quite strong. Thus the best visible indication of strength of rock as a whole is the number and kind of fractures present. In a recent study made by Robert U. King,[9] the kind and the degree of fracturing present was successfully used in mapping cavability of the ore in the mine.

The various kinds of fractures and lines of weakness, based on origin, constitute four groups. The first group is also the oldest and it includes countless small (0.05 to 0.1 in.) quartz-molybdenite veins, some of which have been well cemented with quartz. The second group to develop is made up of many quartz-topaz-pyrite seams carrying minor quantities of chalcopyrite, sphalerite, galena, and wolframite. The third group includes numerous sericite-filled tight joints and open fractures, which cut both molybdenite and pyrite veins. Last, as a result of postmineral movement, many partly cemented fractures were reopened and new fractures formed. As the rock caves, the postmineral fractures and sericite-filled fractures constitute the most favorable surfaces along which separation occurs. Separation along pyrite seams also is a very common occurrence. The quartz-molybdenite veins are the least important lines of weakness.

Ground Movement Observed in Climax Mine

Much of the area that has been caved was not favorable for observation; however, on the whole the ground movement and subsidence in the Climax mine seem to have followed very closely the patterns described in other mines. Vertical tension

fractures at surface are common at distances of several hundred feet outside the limits of caving on the mining levels 400 to 800 ft. below surface; lines drawn in the conventional manner at surface make conventional cave angles as low as 60°. The observed movement does not follow such a cave line and prove it to be fictitious.

Mine openings, favorably placed, made it possible to record in some detail the progress of a caved block and the movement adjacent to it beginning early in 1943 and through 1944 and the greater part of 1945. A representative cross section of the area is shown in Fig. 3. The caved block measures 500 by 500 ft. by 600 ft. high in an ore zone that has a dip of 50°. On the footwall side of the ore zone three raises, spaced 100 to 200 ft., similar to the 150 South service raise shown in Fig. 3, and mine workings at different levels afforded convenient access for observing the nature of the fractures that developed adjacent to the cave block. Thus the points of observation are scattered over a horizontal interval of 600 ft. and a vertical extent of 400 ft., showing that the ground movement is general; not a local feature. The features shown in Fig. 3 are characteristic of the entire area studied.

Mining was initiated in the block as early as 1940, and the block was completely undercut and caved by January of 1943. The details of the initial subsidence at surface were concealed by several feet of snow, but the cracks were evident in April 1943, in spite of the snow. A few months later the surface over the entire caved block had subsided 25 to 50 ft., and vertical tension cracks roughly parallel to the subsided area could be seen on the 30° slope at distances of 150 to 500 ft. from the margins of the main area of subsidence. By midsummer the entire area from the edge of the cave to the most distant crack was intersected by several fractures, and it was evident that the surface over an area of about 400 by 500 ft. was slowly moving down the 30° slope toward and into the main area of subsidence.

The tension cracks formed a series of blocks 50 to 100 ft. wide, 100 to 150 ft. high, and 100 to 500 ft. long. The upper parts of blocks were free to tip forward whereas at depths of 50 to 100 ft. the blocks were not free. Survey stakes were placed on the top of a block at the upslope side of the area in motion, and in 30 days those stakes had moved 3 to 5 ft. parallel to the 30° slope. During the same period no movement was detected in the tunnels immediately below this area.

The movement at the top of the blocks in tipping forward while the base of the blocks was held fast produced additional fractures, until eventually the blocks were reduced to weak piles of rock, which moved down the slope as a rock-slump and rockslide, or a combination of the two.

In rock-slump the surface of a block is reversed as the mass rotates backward at the top and forward at the base. In rockslides the broken material moves in mass, or individual boulders roll on fracture planes parallel to the surface. Rock-slump and rockslide are described by Sharpe[10] as varieties of landslides, which are very common in nature. This type of movement is characteristic of oversteepened slopes in any kind of material. Slides of this type have occurred in the past on the steep slopes of large open pits in the copper mines of the southwest.

In the Denver drift, Grizzly drift, and Buffehr tunnel several cracks formed (Fig. 3). These cracks are approximately parallel to the adjoining caved area, and they dip steeply on both sides of the vertical. These cracks in the mine workings, 150 to 225 ft. below the area known to be moving at surface, were less numerous and they were very narrow (not more than

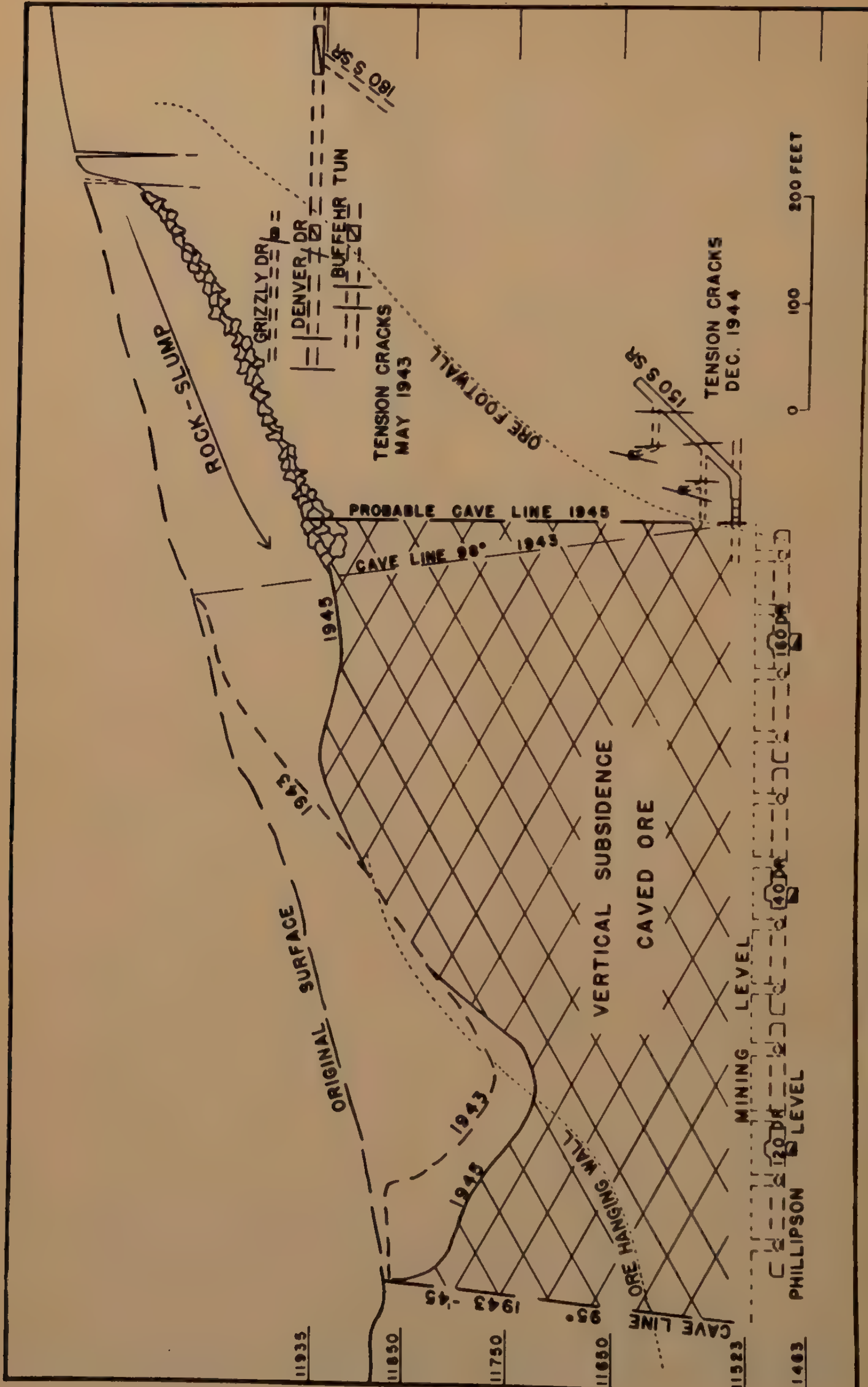

Fig. 3.—Subsidence, tension cracks and rock-slump in the Climax Molybdenum mine.

10 in. wide) as compared with the cracks showing at surface.

The Denver drift is of particular interest because it extended from the edge of the open cave completely across the area that showed movement at surface. Measurement to the open cave established a cave line of 91°. The most distant cracks, 10 ft. or more wide at surface, did not show in the Denver drift 220 ft. vertically below.

The cracks in the mine workings are spaced relatively far apart and the rock between cracks is unfractured, indicating a movement of large units or blocks. In the Denver drift, for example, the four main cracks vary in width from less than 1 in. to 10 in., and they are spaced 20, 30, and 80 ft. The dips of the cracks are from vertical to about 85° away from the caved area. It was not possible to project fractures found on one level of the mine workings to any other level. Also, it was not possible to correlate fractures at the surface with those observed underground. No vertical movement occurred along the cracks and, although the total horizontal displacement represented by open cracks is about 18 in., the intervals between the cracks remained massive without showing new fractures of any kind.

A crack at surface that had opened to 10 ft. by the end of 1943 increased in width to about 40 ft. by the fall of 1944. However, during this time no new cracks formed, and there was no detectable increase in the width of the fractures occurring in the Grizzly drift, Denver drift, and Buffehr tunnel.

In December 1944, new tension cracks were observed in three service raises and related workings, one of which, 150 South service raise, is shown in Fig. 3. The over-all horizontal distance between the several service raises is about 500 ft., indicating that ground movement occurred along the entire width of the caved block. The fractures parallel the caved block, and they are invariably steep, dipping either into or away from the caved area. Widths vary from paper thickness to about 3 in., the maximum widths being nearest the cave. Movement is limited to horizontal displacements, and it was not possible to project fractures any great distance. The tension cracks commonly followed older breaks in rock, but not necessarily the stronger or more conspicuous ones. In this lower area slight movement continued, as indicated by widening (¼ to ½ in.) of old cracks and the appearance of new cracks, from December 1944 to May 1945, after which further movement seemed lacking.

The cracks described are adjacent to the caved area, where the surface slope is about 30°. On the opposite side of the caved area in the bottom of the valley a small crack developed early in 1943 at a distance of about 120 ft. from the edge of the cave. This fracture, in the course of about a year, showed a spreading of 2 or 3 in. and a vertical displacement of about 6 in., after which the crack has remained unchanged, perhaps because the flat surface slope was unfavorable for continued ground movement.

Origin of Tension Cracks

In granite quarries in Vermont, Dale[11] describes offsets of channel holes during operation of granite in shallow quarries. In places the compressive strain is sufficient to shatter the granite. Some quarries show strain in all directions while others only in one direction. Similar conditions are reported in marble and sandstone quarries. Compressive forces of this type are believed to be residual strains that originated in folding or faulting, which formerly was active in the region. Residual forces may be present at Climax also.

The tensile strength of Climax rock

is so small that the relatively sudden release of pressure in one direction, as subsidence over a caved block reaches surface, may be sufficient, without the aid of residual compressive stresses, to produce the tension cracks.

Recapitulation

The significant features and their approximate sequence related to subsidence and ground movement in the block described at Climax are as follows:

1. The initial area of subsidence at surface over the cave block is bounded by cave lines varying from 86° to an overhang of 98°.

2. The vertical tension fractures developed at surface at an early date in a zone 150 to 500 ft. wide adjacent to the subsiding area.

3. The tension fractures decreased downward both in width and in numbers.

4. About 18 months after initial subsidence, small tension cracks developed approximately at the mining level in a zone 100 to 150 ft. wide, adjacent to the caved block.

5. Evidence of low-angle shearing is lacking.

6. The tension cracks at surface formed a series of large blocks or pillars.

7. A mantle of loose rock developed, which moved slowly down the slope in a combination of rock-slump and rockslide.

8. Ore drawn from below removes resistance to the advancing rock-slump and rockslide, and thus further movement is encouraged.

9. The rock-slump and rockslide are limited to a depth of 50 to 100 feet.

10. Subsidence is confined to the area bounded by the initial cave lines.

11. The area in which the vertical tension cracks have been noted roughly corresponds with the area included in the zone bounded by the essentially vertical limit of subsidence and the conventional cave line.

Summary and Conclusions

Ground movement observed in the Climax Molybdenum mine consists of: (1) subsidence in the caved block bounded by essentially vertical cave lines and (2) nearly vertical tension cracks and a combination of rock-slump and rockslide in the zone adjacent to the area of subsidence. The tension cracks decrease in size and numbers with depth, and in cross section the cracks are distributed over a width of about 350 ft. at surface and a width of 100 ft. near the mining level 500 ft. below. The depth of slumping is limited to the interval above the level of the broken rock in the caved block, and the extent of slumping is dependent on the surface slope.

Mine workings at shallow depths would be unsafe, but at levels below the top of the broken rock in the caved blocks the maintenance of tunnels, crosscuts, and other openings might be feasible.

Ore recovery based on the belief that draw is possible on slopes of 60° or even 80° is not to be expected under conditions of successful block caving. The exact slope along which mass subsidence can occur remains to be determined. The smallest angle observed at Climax is 86°, whereas most of the angles are essentially vertical to a 5° or 10° overhanging slope

Ore dilution can be caused under certain conditions by rock-slump or rockslides. However, where the nature of such action is recognized, and with proper draw control, preventive measures should not be difficult.

Direct observations in various mines throughout the country are required for final conclusions, but the writer believes that the mechanics of ground movement and subsidence described will be found to be generally applicable to block caving. Any company interested in mining large tonnages of ore is justified in expending

considerable effort to obtain factual data, as in doing so costly and embarrassing experiences arising from unexpected ground movement may be avoided.

ACKNOWLEDGMENTS

The writer was aided greatly and hereby expresses his appreciation for the discussion of the problems related to ground movement and subsidence of the Climax Molybdenum mine with the several members of the operating staff. Acknowledgment and grateful thanks are due Mr. D. F. Haley, Vice-president of the Company, and Mr. William J. Coulter, General Manager, for encouraging the presentation of the results of the studies. The conclusions and opinions, although influenced by helpful discussions with others, represent the writer's personal views and not necessarily those of the men whose cooperation has been so helpful.

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DISCUSSION

STEPHEN ROYCE*—It is interesting to note from *B*, Fig 2, how closely the subsidence limits approximate to the 60° angle of draw, which is so commonly found to rule in large scale rock subsidence.

The tension cracks shown at *B* are commonly found in the initial stages of subsidence movement and spread like a halo about subsidence limits as movement extends. These tension cracks usually grow into escarpments later on.

Apparently from the illustrations and description given, a system of vertical fractures is the predominating weakness in the wall rock at Climax, and therefore both tension cracks and escarpments are prevailingly vertical in attitude. The illustrations in Mr. Vanderwilt's article seem to conform to the general rule that yield by subsidence is heralded by tension cracks which foreshadow the appearance of the subsidence scarps which are both described in the paper and shown in Fig 2. The difference between the *B* side and the *A* side of Fig 2 is apparently caused mainly by the vertical fracturing which controls subsidence at Climax as compared to the relatively homogeneous rock formation at *A* upon which the original figure in Peele's Handbook was based. The draw at *B* is exactly 60°, while the draw at *A* is a little flatter. The influence of the vertical fractures explains the difference. The surprising feature is the relatively small deviation from the general 60° draw line, which is about the average of rock subsidence experience.

The influence of local fracture systems, faults and bedding in controlling subsidence all the way from extremely low angles of draw to vertical is rather fully discussed in the Peele's Handbook article referred to.

The more illustrations of subsidence there are published the better will be our knowledge. The general principles are pretty well established. The wide local differences reflect the effect of local conditions. This paper is a very valuable contribution to the subject, and for the benefit of the mining industry, it is to be hoped that there will be many more such clear and concise presentations.

Particularly block caving is apt to be of greatly increasing importance in the future of a

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mining industry that must meet high labor costs. More examples of block caving from small scale to large scale operations like those at Climax will serve a vital need.

John W. Vanderwilt (author's reply)—I would like to thank Mr. Royce for his discussion; however, in his comments the meaning of subsidence, draw and tension cracks is not entirely clear. As used in my discussion of subsidence at Climax, subsidence refers to movements involving appreciable downward displacement. Tension cracks indicate only small horizontal separation. "Draw," as observed in the Climax mine, is not a subsidence, but rather a rock failure expressed in the development of tension cracks.

The conclusion by Mr. Royce that vertical escarpments mean "a vertical system of fractures is the predominating weakness in the wall rock at Climax" is not in accordance with facts observed at Climax. Careful and detailed studies of the rock at Climax have failed to reveal a predominating system of fractures that could account for either the vertical subsidence or the tension cracks. In repeating the prevailing belief that predominating fracture systems control subsidence and angle of draw, Mr. Royce overlooks the evidence that gravity is the determining force.

More observations are necessary as to angles of subsidence below levels of the adjoining surface in mines throughout the country; correct explanations can be hoped for only after more facts have been recorded and are better known.

Design of Safe and Economical Arch Structures

By Louis A. Panek,* Junior Member AIME

(New York Meeting, March 1947)

The purpose of this paper is to present a method of designing safe and economical arch structures that are to be constructed of concrete or directly of original earth materials. The experimental data used to illustrate the design method are taken from Bucky and Fentress,[1,2] and the application of the ideas presented by them is extended by obtaining an additional set of curves from which arch structures may be designed.

This design method is based upon the following principles of similitude: A scalar model made of the same material as that of the full sized structure will behave similarly to its prototype if the density of the model is increased in the same proportion as its linear scale is decreased. Let R represent the scale or model ratio, namely, the ratio of a prototype linear dimension to the corresponding model linear dimension. The effect of increased density may be obtained by rotating the model in a centrifuge at a speed such that the centrifugal field exerts on the model an accelerative force R times the accelerative force of gravity. Under these conditions; (1) model unit stresses are equal to corresponding prototype unit stresses, (2) the ratio of the prototype deformation at a given point to the model deformation at a similar point is equal to the scale ratio, R, and (3) the model and the prototype will behave similarly within and beyond the elastic limits of strain.

Manuscript received at the office of the Institute Jan. 23, 1947. Issued as TP 2266 in Mining Technology, November 1947.

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[1] References are at the end of the paper.

According to the above relations, such models can be used to solve the following problems:

1. The dimensions of the most economical safe arch structure for a given load.
2. The maximum safe roof span underground for given geologic conditions.

The procedure is described briefly as follows. Tests are made by rotating in a centrifuge a number of models that represent a variety of prototype loads and dimensions. From the experimental data, curves are plotted to show the relations between these loads and dimensions. The curves are used to solve directly the problems stated above.

Structural design is based upon the working (safe) stress of the material of construction, which is established by applying a factor of safety to the experimentally determined ultimate strength of the material. In conformity with this principle, each model is gradually loaded until failure occurs. A safety factor is then applied to the results. If a safety factor of 4 is used, a given model at failure represents a safe prototype, the linear dimensions of which are equal to one-fourth the model ratio at failure times the corresponding model dimensions. The safe model ratio is computed from the following formula:

$$\text{Safe } R = \left(\frac{1}{4}\right)\frac{4\pi^2n^2r}{g} = \frac{\pi^2n^2r}{g} \qquad [1]$$

in which: n = centrifuge revolutions per second at model failure;

r = radius of rotation of the model in feet;

g = 32.2 ft/sec^2.

Design Procedure

The experimental data are given in Table 1. The testing equipment has been described by Bucky and Fentress.[1,2] The data. For other than a segmental arch, it might be satisfactory to use span, rise, and total depth in Fig 3. The writer has not tested any but segmental arches, and with-

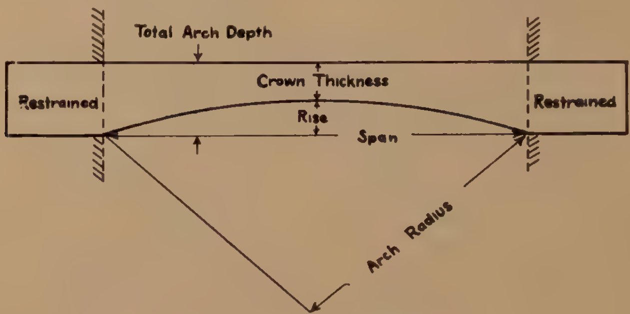

Fig 1—Shape of arch tested.

shape of arch tested is shown in Fig 1. It may be described as a fully restrained beam from the under side of which a circular segment has been removed, the length of the chord being equal to the span. The design method is equally applicable to other shapes of arches. It is considered that the following general procedure may be utilized to obtain conveniently direct solutions to many support problems without recourse to complicated design formulas: (1) make tests by rotating in a centrifuge a series of models that represent a variety of prototype loads and dimensions, (2) from the experimental data, plot curves showing the relations between these loads and dimensions, and (3) solve the design problem directly from the curves. The segmental arch discussed has five variable dimensions: the *span* and the *arch radius* together determine the *rise*; the *rise* plus the *crown thickness* equals the *total arch depth*. Clearly, only three of these are independent variables, thus a segmental arch is completely specified by (1) span, radius, and total depth (as in Fig 3), or (2) span, rise, and crown thickness, or whichever other combination is most convenient to use or will most clearly show the relations brought to light by a study of the experimental out experimental data for other types of arches a definite statement cannot be made. The general design method presented is possible of wide application; the specific illustration of that method is believed to be eminently suitable for the design of segmental arches submitted to body loads.

A series of models of constant span and arch radius and of variable total depth of arch are prepared and set in holders. Each model is rotated in a centrifuge until failure occurs. For each model the safe model ratio is computed from Eq 1. The model linear dimensions are then multiplied by the safe model ratio to determine the corresponding safe prototype linear dimensions. The data thus obtained (Table 1) provide points on a curve, such as Curve A, Fig 2, which shows for a 20-in. model arch radius the relations between the prototype safe total depth of arch and the prototype safe span and the safe model ratio. Next, tests are made using a series of models having the same span as before, a constant but different arch radius, and a variable total depth of arch. This procedure is repeated in order to obtain a group of curves as shown in Fig 2.

The data required to construct the arch design curves (Fig 3) are obtained as fol-

TABLE 1—*Experimental Data*

Model Dimensions, In.						Radius of Rotation, Ft	RPS at Failure	Safe Model Ratio ($SF = 4$)	Safe Prototype Dimensions, Ft				
Number	Arch Radius	Crown Thickness	Rise	Tota Arch Depth	Span				Arch Radius	Crown Thickness	Rise	Total Arch Depth	Span
1	20	1/8	0.273	0.398	6	0.688	25.2	134	231	1.4	3.1	4.6	74
2	20	3/16	0.273	0.461	6	0.700	30.8	204	339	3.2	4.6	7.8	110
3	20	1/4	0.273	0.523	6	0.710	33.3	242	402	5.0	5.5	10.5	131
4	30	1/8	0.178	0.303	6	0.715	14.3	45	113	0.5	0.7	1.1	24
5	30	1/4	0.178	0.428	6	0.737	27.0	159	398	3.3	2.4	5.6	86
6	30	3/8	0.178	0.553	6	0.751	31.1	209	521	6.5	3.1	9.6	112
7	40	1/8	0.131	0.256	6	0.708	11.4	28	94	0.3	0.3	0.6	15
8	40	1/4	0.131	0.381	6	0.713	20.4	91	302	1.9	1.0	2.9	49
9	40	3/8	0.131	0.506	6	0.714	27.8	169	562	5.2	1.8	7.1	91
10	120	1/4	0.045	0.295	6	0.708	11.8	30	300	0.6	0.1	0.7	16
11	120	3/8	0.045	0.420	6	0.712	21.6	102	1,020	3.2	0.4	3.6	55
12	120	1/2	0.045	0.545	6	0.709	28.7	179	1,790	7.4	0.7	8.1	97
13	∞	0.302	0	0.302	6	0.698	11.6	29	∞	0.7	0	0.7	16
14	∞	0.428	0	0.428	6	0.715	20.4	91	∞	3.2	0	3.2	49
15	∞	0.552	0	0.552	6	0.711	27.6	166	∞	7.6	0	7.6	90

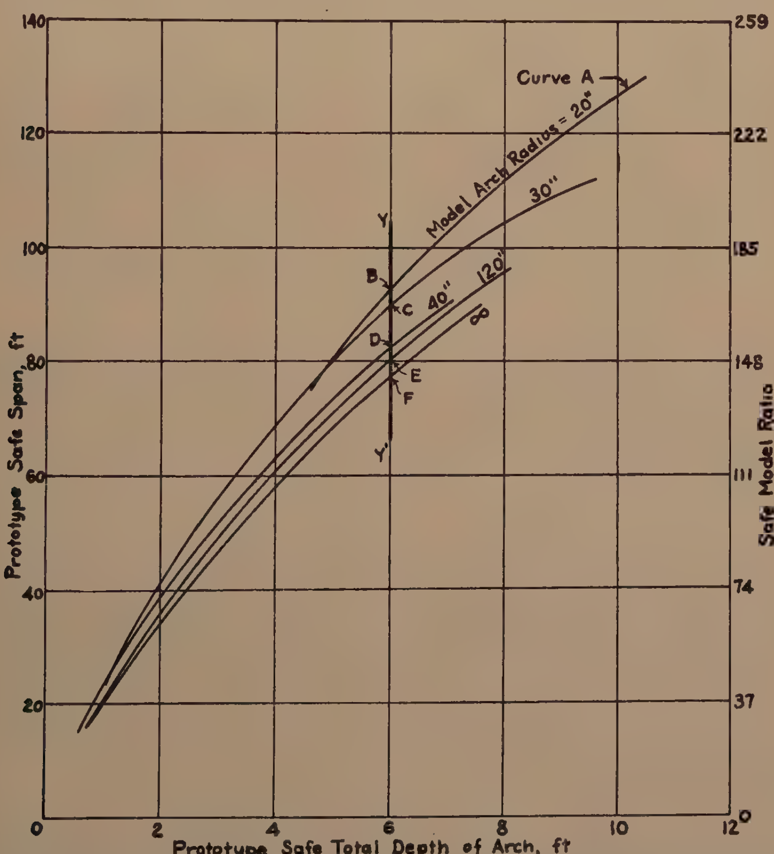

FIG 2—RELATION BETWEEN PROTOTYPE SAFE TOTAL DEPTH OF ARCH AND PROTOTYPE SAFE SPAN AND SAFE MODEL RATIO FOR EACH MODEL ARCH RADIUS.

lows. Draw line YY′, Fig 2, in order to determine the relation between the prototype arch radius and the prototype safe span for a 6-ft prototype total depth of arch. At the point of intersection of YY′ with the 20-in. model arch radius curve (B, Fig 2), the prototype safe span is 93 ft, and the safe model ratio is 172; therefore, the corresponding prototype arch radius is equal to 172(20/12) = 286 ft. In the same manner, the prototype safe span and the prototype arch radius are determined at the points of intersection of YY′ with the other model arch radius curves (C, D, E, F). The curve *total depth* = 6 ft, Fig 3, is obtained by plotting the above determined values of prototype safe span v. the corresponding prototype arch radius. This procedure is repeated in order to obtain the curves for various other prototype safe total depths of arch (2 to 7 ft). Referring to Fig 3, note that:

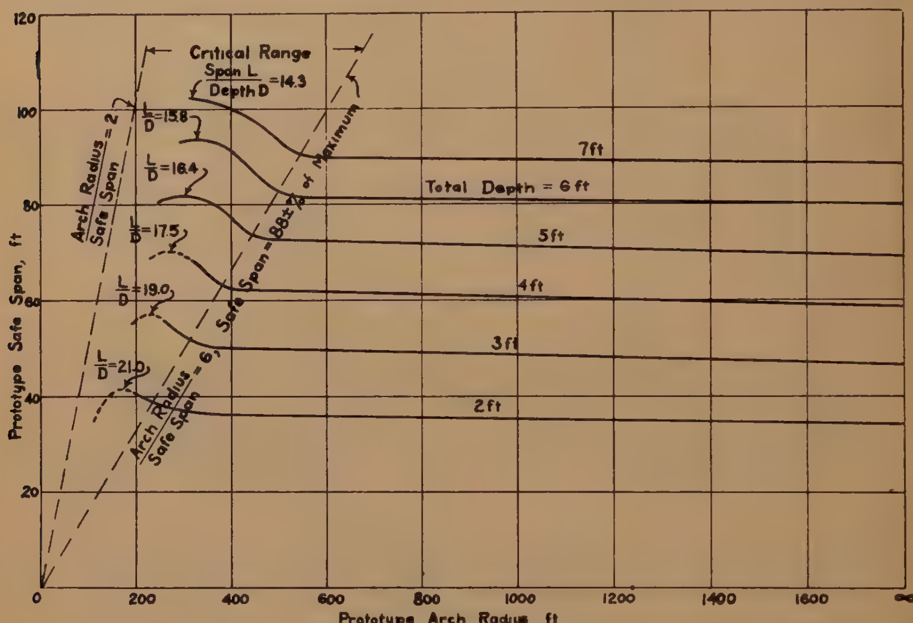

FIG 3—ARCH DESIGN CURVES.

1. For a given prototype total depth of arch: (a) there is a critical value of arch radius that permits the maximum safe span; (b) the safe span is decreased appreciably by increasing or decreasing the arch radius from the critical value, within the limits (arch radius/safe span) = 2 to 6; (c) the safe span is negligibly decreased by increasing the arch radius, if (arch radius/safe span) > 6.

2. For a given prototype safe span: (a) there is a critical value of arch radius that requires the minimum total depth of arch; (b) the required total depth of arch is increased appreciably by increasing or decreasing the arch radius from the critical value, within the limits (arch radius/safe span) = 2 to 6; (c) the required total depth of arch is negligibly increased by increasing the arch radius, if (arch radius/safe span) > 6.

Fig 3 can be used to design arch structures in which the strains are caused solely by the body load, or in which the strains caused by an externally applied load are negligible compared with those caused by the body

load. For example, assume that an 80-ft span is desired. If a 600-ft arch radius is used, the required safe total depth of arch is 5.8 ft; whereas with a 290-ft arch radius, the minimum required safe total depth of arch, 4.8 ft, can be used, thus effecting a considerable saving of material. Assume that an 80-ft thick vertical deposit is being worked by an open or shrinkage stope method of mining. If the under surface of the level pillar is flat (arch radius = infinity), the required safe total depth (level pillar thickness) is 6.4 ft; whereas by arching the under surface of the level pillar to a 290-ft radius, its safe total depth need be only 4.8 ft, thus increasing the amount of ore extracted. In a similar manner, if the safe total depth of arch, say 5 ft, is specified, Fig 3 shows that the maximum span, 82 ft, is obtained by using a 300-ft arch radius.

The following additional facts are obtained from Fig 3:

1. For any given value of total depth of arch, if one chooses a safe span for which (arch radius/safe span) = 6, this span is 88 pct of the maximum possible safe span for the total depth specified.

2. Clearly, the greater the ratio of the safe span to safe total depth, the more economical the structure. In Fig 3, however, note that as the desired safe span becomes greater, the ratio of the safe span to the safe total depth decreases. Compare the points of maximum safe span on the total depth curves. If a 42-ft safe span is desired, the ratio of the safe span to the safe total depth is 21.0; if an 82-ft safe span is desired, the ratio of the safe span to the safe total depth is 16.4. For a concrete structure this indicates that an economic balance exists between constructing one long span with two supports, and two shorter spans with three supports. In an underground mining operation, if both the pillars and the roof are in ore, a similar economic balance exists. If the roof is in waste, however, the ratio of the safe span to the safe total depth is not economically important, in which case it may be preferable to use the maximum possible safe span.

The method described above may also be applied to the design of an arch, the strains in which are the result both of external and body loads. Such an arch may be a bridge, in which case it is possible to investigate the effect of concentrated loads at various points on the span; it may be a mine pillar subjected to a load caused by fill or broken ore in the stope above; it may be a concrete arch required to support a weak roof in a tunnel or other opening underground. Thus the most economical structure can be determined for any loading condition of the above types.

Acknowledgment

The writer wishes to thank Prof. Philip B. Bucky for reading the manuscript and for making many helpful suggestions. His generous advice assisted greatly in the preparation of this paper.

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Industrial Minerals of Ethiopia

By Thomas G. Murdock,* Membfr AIME

(Atlanta Meeting, October 1947)

Ethiopia, the first country to be liberated from Axis domination, has recovered remarkably from the ravages of occupation and war. Mineral production has contributed significantly towards this recovery, as it has provided a means of obtaining foreign exchange and has given employment to hundreds of workmen whose maintenance otherwise would have presented a problem. Although primarily an agricultural country, never with a large mineral production, Ethiopia has within its borders a variety of minerals, both precious metals and nonmetallics. Some of these are important to the country as a national asset and others contribute greatly to local economy and can be expected to do so even to a greater extent in the future[1] (Fig 1).

Physiography and Geology

The Empire of Ethiopia is essentially a mountainous and volcanic country, in northeast Africa, situated principally between latitude 4° and 15°N and longitude 34° and 44°E. It is bounded on the north by Eritrea, on the northeast by French Somaliland, on the southeast by Italian Somaliland, on the south by Kenya Colony, and on the west by Anglo-Egyptian Sudan. These European possessions thus completely surround the country. Its access to the Red Sea is through Eritrea; to the Gulf of Aden, through French or British Somaliland; and, to the Indian Ocean, via Italian Somaliland. These routes range in length from 40 to 250 miles. Ethiopia has an area of approximately 350,000 square miles, with a width from north to south of 625 miles, and a maximum length from east to west, along latitude 8°N, of about 875 miles.

Physiographically the country is characterized by two very extensive sub-tabular plateaus: the Ethiopian Plateau to the northwest and the Somaliland Plateau to the southeast, separated by a long tectonic graben—the Rift Valley of the Galla Lakes and Danakil. The Ethiopian Plateau, deeply cut and notched by the drainage, slopes in a general northwest direction toward the Sudanese Plains and the valley of the Nile; the Somaliland Plateau slopes to the southeast, toward the Indian Ocean. The valley of the Galla Lakes and its extension to the northeast—the Awash River Valley, separates the plateaus, from Lake Rudolph to Danakil. This depression which constitutes a continuation or ramification of the Great Rift Valley of Central Africa joins in turn the trough of the Gulf of Aden and the Danakil-Eritrean Rift, continuations of the Red Sea and the Jordan Valley.

The general basement of Ethiopia is made up of very old rocks, usually considered as Archaen.[2] These rocks are represented by the metamorphic crystallines, such as gneisses, different kinds of schists, included lenses of crystalline limestone, and massive intrusives such as granites, diorites, and syenites, all cut by

Manuscript received at the office of the Institute Sept. 26, 1947. Issued as TP 2356 in Mining Technology, March 1948.

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[1] References are at the end of the paper.

FIG 1—PRINCIPAL INDUSTRIAL MINERAL DEPOSITS OF ETHIOPIA.

1,2, Salt deposits of Danakil. 3, Mt. Dallol potash deposit. 4, Mt. Dofan sulphur deposit. 5, Magado Salt Lake. 6, El Sod Salt Lake. 7, Dire Dawa cement plant and quarry. 8, Mt. Achim limestone quarry. 9, Negelli limestone quarry. 10, Takazzee limestone deposits. 11, Addis Alem lime kilns. 12,13,14,15,16 brick plants. 17, Yekka pottery center. 18, Ambo monumental stone quarry. 19,20,21,22, Crushed stone quarries. 23, Mt. Jueppi, asbestos. 24, Dawale gypsum deposits. 25, Addisgie alabaster deposit. 26, Shebelli mica mine. 27, Karara mica mine. 28,29, pumice deposits.

dike-like intrusions. Upon the eroded surface of the crystallines, sedimentary rocks of Mesozoic age are to be found, except where these have been removed by erosion or covered by later lava flows. The oldest of these Mesozoic sediments is Triassic sandstone, normally quartzose and white, but also red and variegated. This formation is in part continental and in part marine, having been laid down in shallow seas and in lagoons, and often contains gypsum intercalations. It is followed by Jurassic crystalline limestones and marl of a thickness of over 1000 ft. The Jurassic in some places is followed by a sandy facies of the Lower Cretaceous and some marine Upper Cretaceous. In certain marginal areas the Tertiary is represented by marine or terrestrial sedimentary rocks. The Mesozoic or Tertiary sediments, and in some places the Archaen basement, are overlain by a thick series of volcanic flows, essentially composed of basalts and trachytes, of Tertiary and Quaternary age.

Industrial Minerals and Rocks

The Salines and Sulphur

Salt—Although salt is produced in Ethiopia, over half of the country's requirements is met by imports from adjacent territories where it is obtained mainly by the solar evaporation of sea water in artificial basins. The largest deposits in Ethiopia are in the Danakil depression—a vast sunken zone which reaches a depth of 392 ft below sea level and has an area of about 3000 square miles. It is estimated[3] that of this area about 460 square miles are actually covered by salt, and the reserve is estimated at over one billion tons. The salt, accompanied by gypsum, was formed by the evaporation of the water of the closed basin that was formed from the elevation of the so-called "Danakil Alps," which are between the depression and the Red Sea. Exploitation of the deposit has been by the Danakil inhabitants, utilizing hand methods, and production is believed to average about 10,000 tons annually. The total consumption of the country has been placed at about 28,000 tons. The construction of a truck road from the Salt Plain of Danakil to the Plateau in Tigre Province, is clearly needed to replace camel transportation and thus facilitate a larger production and delivery, making available a greater salt supply at a lower price to the ultimate consumer.

There are other occurrences of salt in Ethiopia; the most important are the saline lakes Magado and El Sod, near Mega, in southern Ethiopia. These occupy craters of extinct volcanoes. Both of these lakes are exploited on a small scale by the local inhabitants (Fig 2). The brine from this lake is a mixture of sodium chloride and carbonates. Based on a brief reconnaissance survey these lakes appear quite valuable as a source of salt and carbonates for local use; there is a possibility that further investigation might indicate the feasibility of the small-scale utilization of the Magado brine for soda ash manufacture.

Potash—Deposits of the potassium minerals, sylvite and carnallite, have been exploited at Mt. Dallol on the Salt Plain of the Danakil depression, 51 miles south of the Red Sea port of Mersa Fatma. The occurrence bears a certain resemblance to the famous Stassfurt deposits in Germany.[4] There is no record of activity there since 1933. The 1927 production is reported at 2500 tons, and exports are said to have been previously made to Italy, Great Britain, Egypt and Japan. A carnallite zone surrounds the area of sylvite and has a potassium chloride content of 15 to 25 pct. It is possible, without a special plant, to obtain an 80 pct KCl product by the simple process of exposing the material extracted to the hot and humid atmosphere.[5] The reserves have been estimated at 80,000 metric tons of mixed salts of which 50,000 would be commercial-grade potassium chloride. This figure is for the immediate

environs of Dallol and does not consider adjacent areas.[3]

There are unverified reports of the occurrence of potassium nitrate in the vicinity of Fiche and Debra Libanos in Shoa Province, and northeast of Jimma in Kaffa Jimma.

Sulphur—Several occurrences of sulphur in the volcanic regions of the Awash Valley are known, however, the only one which is near transportation is that at Mt. Dofan, 45 miles from Awash Station on the railway which connects Addis Ababa, the Ethiopian capital, with the port of Jibuti in French Somaliland. At Mt. Dofan the sulphur occurs in fumarolitic zones around a part of the perimeter of a volcanic crater, filling the pores and fractures of a weathered trachyte (Fig 3). Although there is little upon which to base a reserve estimate it seems that there might be 150,000 tons above the crater floor. This is obviously low-grade material. Mining, concentration and transportation would present no serious technical difficulties. The deposit has not been prospected and until this is done it is not possible to venture an opinion as to its commercial possibilities.

FIG 2—SMALL-SCALE SALT MINING, LAKE EL SOD.

Limestones

The importance of the limestone deposits of Ethiopia is reflected in the fact that they have already provided the basic raw material for a cement plant at Dire Dawa, as well as a useful construction material in general. The limestones occur both as intercalations in the Archaen crystalline basement rocks and as typical sedimentary deposits of Jurassic and Cretaceous age. Thus the limestones have a fair distribution in the country. Although the road construction program utilized limestone wherever a road was passing through areas of this rock, the principal exploitation has been at Dire Dawa, with secondary development near Harar, Negelli, and in Tigre. The Dire Dawa quarry is a model installation with an extraction system for glory hole mining of the limestone into raises connecting with underground haulage drifts. However, during recent years a shortage of explosives greatly changed the normal functioning of this method and instead a large force of workmen has been engaged in laboriously breaking up by hand the float material and outcropping ledges,

thus producing about 80 tons per day. On Mt. Achim, near Harar, limestone is produced for building and crushed stone; it was formerly burnt for lime. Similar operations were carried out near Negelli and it was reported that an annual average of 16,000 metric tons of slack lime was produced during the Italian occupation at Addis Alem, west of Addis Ababa.

FIG 3—OCCURRENCE OF SULPHUR IN WEATHERED TRACHYTE, MT. DOFAN.

Clays

The principal utilization of clays in Ethiopia has been for the manufacture of brick and other structural clay products and for making native pottery. Both the soils and the clays of the country are essentially residual, however, in some localities sedimentary clays have been formed by fluvial or lacustrine deposition. Clays suitable for brick making are found in various localities and during the Italian occupation modern brick plants were operated at Harar, Addis Ababa and Jimma; there is an unverified report of the production of 300,000 brick per month at Lekemti, in Wollega Province. The sedimentary clay of the Jimma Basin provided a satisfactory material for the operations there, and the red and yellow clays of the crystalline area of Harar seem to be even better suited. The flood-plain clay at Gambela, at the head of wet-season navigation on the Baro River, was being utilized on a small scale in the early part of 1945.

There are no known deposits of a true kaolin in the country; the pegmatite dikes are commonly unkaolinized. However, in many areas there are residual clays, the product of the kaolinization of the feldspar contained in the original rock (Fig 4). In fact many occurrences of a laterized trachyte are known throughout the country and present a great similarity, at least in appearance, to true kaolin deposits, except for a high iron content. They are utilized in many localities for the manufacture of water jugs and other containers of various sizes, by blending with other clays to provide plasticity and a better firing body. At some places, as around Yekka near Addis Ababa, a small local home industry has been developed and the requirements of a large area are produced. Straw and cattle droppings are the fuel for this small ceramic industry. All of the pottery is formed by hand without the use of a potters wheel.

Other Construction Materials

The rocks of the different formations of Ethiopia have found a wide application throughout the country as building stones and crushed rock. The utilization of these has generally been influenced by transportation (Fig 5). For example, the granites are perhaps the most attractive of any of the rocks, yet their use in public buildings is practically unknown as they are not found immediately adjacent to the larger cities. The trachytes, traps, sandstones and limestones have all provided a source of building stone. For highway construction the rock encountered along the right-of-way and in cuts has been used for fills and structures. Many small roadside quarries are to be seen along the highways, where small portable crushers were used to prepare the base and surfacing material. Some of the highway structures are truly imposing ones, particularly along the mountain roads where huge masonry piers, arches, and retaining walls afford both utility and a pleasing appearance. It is believed that, in general, little attention has been given to the selection of material used for crushed stone; weathered rock, if available, has been used.

In addition to the smaller roadside quarries, larger ones have been opened around Addis Ababa, Harar, Gondar, Dessie, Ambo, Negelli and Jimma (Fig 6). A trachyte which closely resembles sandstone is quarried at Ambo, west of Addis Ababa, and this material, known as "Ambo stone," is widely used for monumental and ornamental construction. It works easily but is said to wear very poorly.

The low cost of labor has permitted a rather extensive use of masonry and consequently less brick and cement construction has been used than would be expected. The ease with which the trachytes and columnar-jointed basalts can be quarried and shaped is likewise a favorable factor. The working of stone has been entirely by hand methods. The wide use of thatched straw and similar primitive construction in the rural areas has limited the use of building stone to the cities and larger towns.

FIG 4—KAOLINIZED GRANITIC ROCKS NEAR ARERO, SOUTHERN ETHIOPIA.

Miscellaneous Industrial Minerals and Rocks

The occurrence of a few other nonmetallics in Ethiopia is important as some of these would have a definite place in any industrial development. It is doubtful if any exports could be made, with perhaps one or two exceptions. A definite program is needed to investigate the reported occurrences of the non-metallics not yet verified, to prospect those which might have possibilities, to carry out the necessary research

upon the technology of the materials, and to study the markets and their possible development.

Millstones, usually made of trachyte or basalt, are widely used in Ethiopia. There are also sources of materials which will serve for making grindstones and powdered abrasives.

Anthophyllite and vermiculite occur in a serpentinized mass of basic rock on Mt. Jueppi, east of Harar, and there has been some production of the former (Fig 7). The limited amount of prospecting there does not permit a determination of the quantities which might be produced but the area appears worthy of exploration.

5 

6 

Fig 5—Primitive transport of basalt, Akaki River.
Fig 6—Basalt quarry, Addis Ababa.

Feldspar occurs quite widely in the area of crystalline rocks between Harar and Jigjigga. It is mainly of the potash variety and a cheap product could be obtained by merely gathering up the loose pieces covering a large area southeast of Dire Dawa.

Gypsum occurs rather extensively on the Danakil Salt Plain, in association with the salt; however, transportation difficulties have hindered its exploitation. The main source of the country is from Dawale, on the railroad between Dire Dawa and Jibuti.

The production is believed to be about 1000 tons per year and the chief use is as a retarding agent in cement. At Dire Dawa, gypsum is calcined separately and sold as plaster of paris. An alabaster variety occurs as a replacement in limestone in the Fiche area of Shoa Province and is used for ornamental carving. Improvement of transportation and any additional cement production would permit a wider development of the gypsum resources.

Mica has been produced at the Shebelli and Karara mines near Jigjigga and there are other occurrences in the pegmatite dikes throughout the areas of crystalline rocks. These dikes are usually unkaolinized and the mica content is generally very low; also the amount present is of small size and low quality. It remains to be proved that commercial mica can be profitably produced in Ethiopia.

Pumice beds are found south of Ada Station on the railroad and along the highway between Mogjio and Awash. A small quarry was opened at the former locality during the Italian occupation and the output was apparently used as a lightweight aggregate.

There are references in the literature to the occurrence of magnesite, graphite, phosphates, precious stones, and talc, but these lack verification.

Quartz occurs fairly well distributed throughout the areas of crystalline rocks, both in pegmatite dikes and in quartz veins. No occurrence of the strategic crystal variety is known.

Mineral Development

In Ethiopia, as in other parts of the world, the main emphasis has been on the precious metals. The first reference to the gold of the country is in the Holy Bible. Although some of the less spectacular nonmetallics, such as the clays and salt, have found a utilization for centuries, the exploitation of these as industrial raw materials is still on a small scale and was, for the most part, inaugurated during the brief Italian occupation.

Unfortunately, lack of transportation and other factors hindered reconaissance

Fig 7—Anthophyllite produced from serpentine mass forming Mt. Jueppi near Harar.

and prospecting for a long time. A chain of circumstances combined to delay an authoritative appraisal of the possibility of developing an active mineral production industry in the country as a whole.

A single-track, meter-gauge railroad, 500 miles long, links the Ethiopian capital with Jibuti. A few improved highways—inadequately maintained, but overdesigned for the traffic they are now carrying—extend like spokes of a wheel to connect the capital with the outlying smaller cities, and comprise the transportation net.[6] One of these highways from Addis Ababa to Assab is approximately 450 miles long, and another from the capital to Massana is approximately 550 miles long.

The proven fuel resources of the country consist of a limited supply of wood, and some small undeveloped lignite deposits. A concession has recently been granted to an American concern for the petroleum rights.

The discovery of the occurrence of oil would greatly change the future development of the country in many respects. Although such a discovery is possible, there has been no disclosure thus far that would make it seem imminently probable.[6]

The Ethiopians have stated that they are eager to join with the peoples of the United Nations "in their magnificient effort to rebuild a war-torn world." They recognize that the first step in this direction is to rebuild and develop their own battle-scarred country and to raise their own standard of living. They recognize the importance of extending their cultural boundaries to make greater use of science, mechanics and the arts. There is need of much that the outside world can give; outside influences having to do with the industrialization and scientific education that are now being invited by the Government of Ethiopia.[6]

An official interest in mineral development has been definitely assured by the creation of a geological department and negotiations with several American companies to undertake reconnaissance, exploration and development of mineral deposits. It seems possible that the application of modern geological and technological methods to the industrial mineral field in Ethiopia might be productive; it will do much toward bringing about the benefits enjoyed by other countries through exploitation of this most interesting and useful group of minerals.

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Barite Production in the United States

By Albert C. Harding,* Member AIME

(Los Angeles Meeting, October 1946)

For several years barite ($BaSO_4$) production has topped such better known minerals as feldspar and fluorspar in annual tonnage and is now well established among our foremost nonmetallic industries.

Description

Of widespread occurrence in the United States, barite is relatively soft (hardness, 2.5 to 3.5), inert, heavy (specific gravity, 4.5), and is also called "barytes" (sometimes pronounced "bear-i-tease"), "heavy spar" and "tiff." It occurs with granular and earthy structure but usually has coarse, orthorhombic crystallization. It is commonly white, and although occurring in various shades of red, yellow, and gray, barite always gives a white streak and can be scratched by the ordinary prospecting pick. Its softness and weight are usually sufficient to establish field identification. Another aid is the "rotten egg" odor of H_2S which sometimes emanates from freshly fractured surfaces.

Barite can be readily distinguished from two other minerals which have somewhat similar occurrences and characteristics. The relatively rare celestite, strontium sulphate, has a specific gravity slightly less than 4.00 and gives a distinctively crimson color in a flame test. The very rare witherite, barium carbonate, has a specific gravity of 4.3 but readily effervesces with dilute hydrochloric acid.

Occurrence

All known commercially important barite deposits in the United States occur as replacements or cavity fillings in schists, shales, limestones, dolomites, and quartzites, or as residual deposits from such formations. In Georgia, Tennessee, North Carolina, and Missouri, high-grade barite is recovered as fragments from residual clays covering quartzites and dolomites. In Arkansas, Nevada, and California, barite, usually of lower grade, is mined from replacement deposits. These seven states contributed the entire production reported by the U. S. Bureau of Mines for 1945 as given in Table 1.

Table 1—*Crude Barite Sold or Used by Producers in 1945*

State	Short Tons
Arkansas	260,660
Missouri	225,467
Georgia	110,393
Tennessee	32,812
Nevada	28,919
Other States	37,811
Total	696,062

"Other States" included only California and North Carolina. Such a large part of this undistributed production came from California that that state probably outranked both Nevada and Tennessee. For the second consecutive year Arkansas was the leading producing state, particularly worthy of note because there was no commercial production of barite there prior to 1941 and because the entire production of Arkansas has come from one large deposit which is being operated by two companies.

Consumption

The 1945 consumption as reported by the Bureau of Mines was:

482,442 tons as ground barite
139,288 tons in making lithopone
99,173 tons in producing barium chemicals
———
Total 720,903 tons

Manuscript received at the office of the Institute Dec. 12, 1946. Issued as TP 2414 in Mining Technology, July 1948.

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The Bureau further reported that 87 pct of the ground barite (58 pct of the total) was used as weighting material for oil-well drilling muds. The use of ground barite for this purpose has accounted for most of the increased production in recent years. The tonnage used in oil wells in 1945 was just about twice the tonnage of ground barite consumed for all purposes in 1943. Other uses in 1945 were in manufacturing the following commodities: glass, 6 pct; paint, 4 pct; rubber, 2 pct; and undistributed, 1 pct.

Specifications

Specifications for most of the ground barite used in oil wells are 95 pct minus 325 mesh, 4.30 specific gravity, and low mud viscosities. California production usually has slightly lower specific gravity than ground barite from Arkansas, Missouri, and Georgia, but the other requirements are the same. Ground barite is usually sold in carload lots, packed in 100-lb net weight, 4-ply, paper bags. The average value for 1945, reported by the Bureau of Mines, was $16 a short ton.

Most of the barite consumed for purposes other than drilling muds is either mined by the consumer or is purchased unground as washed, crushed, and jigged barite. The latter is known as "chemical grade" and specifications are usually in terms of $BaSO_4$ and iron content. In Missouri this material is now sold f.o.b. mines for $8.50 a short ton with 94 pct $BaSO_4$ and 1 pct iron with bonuses and penalties of $0.25 for each percentage of $BaSO_4$ plus and minus 94 pct. Purchase contracts show considerable variation in specifications. While in some cases penalties are provided for excess iron, contracts usually guarantee that the iron will not exceed a stipulated maximum content. In Georgia similar specifications prevail at $9 per long ton. It is reported that some of the Georgian barite is now being treated by magnetic separation to remove iron.

The Office of Price Administration allowed two Missouri producers a selling price of $11.50 a ton for glass-grade ore in May and June 1946. On July 26, barite was exempted from all price controls but there has been no appreciable change in the price structure by this time.[a]

Statistics regarding exports are not available but some material was shipped out of this country for use in drilling muds, and 11,576 tons were exported as lithopone, a precipitated material containing 70 pct barium sulphate and 30 pct zinc sulphide, used as a pigment and filler in paint and rubber.

Large deposits are known to exist in Canada and Mexico, and, although our productive capacity is ample for present requirements, several low cost foreign producers are in strategic locations with respect to transportation costs, and in 1945 we imported 56,894 tons of crude barite. Very little ground barite is imported because the domestic industry is partly protected by a duty of $7.50 a ton on this material.

Mining and Milling

Probably 95 pct of our current barite production comes from surface operations.

Missouri, Georgia, and Tennessee

Mining methods and treatment in the Cartersville District, Ga. and the Sweetwater District, Tenn. are similar to those employed in Missouri, the second largest producing state. The industry in Missouri is fairly well concentrated in Washington County, southwest of St. Louis. Here the barite occurs as fragments with chert and quartz in residual clay deposits which have weathered from dolomites. Former hand-mining methods in which small vertical shafts were sunk and then chambered in the best tiff-bearing strata have now given way to large scale operations in which

[a] December 1946.

tractors, power shovels and dump trucks are standard equipment. While some selectivity is practiced, particularly in wasting barren or low-grade top soil, most of the clay containing barite fragments, from sand size up to about 2 cu ft, is now mined and hauled to nearby washing plants. The clay is dumped into a hopper or "bull pen" where it is hit with hydraulic water jets and washed onto stationary or rotating rail grizzlies. Usually some sorting is done at this point. Unbroken lumps of clay and waste rock are thrown aside and large pieces of barite are hammered through the rails, usually set at about 4 in. The slurried material then passes through double log washers where the dirt and clay are removed and overflowed to mud ponds. The log discharges washed "ore" and waste rock to trommel screens or picking belts. Some further sorting usually is done here and at some washers coarse galena is hand picked from the belts. Trommel oversize, about ⅜ in., is crushed and all the material is jigged to make a high-grade concentrate.

The jigged material may be shipped directly to consumers or it may be sold to another producer who maintains a grinding plant and also blends and segregates the jigged tiff from several washers to meet exacting consumer specifications for "chemical grades."

No additional concentrating is done at the grinding plants except that which occurs in dewatering tanks after the material is ground in ball mills to 95 pct minus 325. The ground barite is dried on rotary steam-film driers and then packed in 100-lb paper bags for carload shipments.

Arkansas

Magnet Cove, Ark., is now the most productive barite district in the United States. Present production comes from one large deposit near Malvern, Hot Springs County. Two producers are working opposite ends of the deposit which occurs in a large synclinal fold, known locally as the Chamberlain syncline. In plan the outcrop is approximately a *V*, closed at the east end and diverging to the west. Each side of the *V* is almost a half mile long and the two sides are about a quarter mile apart at the widest point. The axis of the synclinal fold plunges about 18° to the west. Any cross section perpendicular to the axis is approximately *U* shaped with the *U* getting larger as the sections proceed westerly until it is over 600 ft high and a quarter of a mile wide at the biggest section.

The barite is a deposition in shale at the base of the Stanley shale series and rests on a base of hard Arkansas novaculite. The overburden, above the barite and within the *U*, is a mixture of shales, sandstones, and clays. The south limb dips almost vertically while the north limb is flatter, dipping about 22°. Extensive core drilling has delineated the size and shape of the deposit although all mining operations so far have been within 50 ft of the surface. The ore zone varies up to 70 ft in thickness but the commercial zone will average from 30 to 40 ft. The deposit is low-grade barite, running from 60 to 70 pct $BaSO_4$.

Operations so far have consisted solely of open-cut methods, removing part of the overburden from the hanging wall within the *U*, by scrapers and dump trucks and depositing the waste around the outside edges of the pit. This method poses economic limitations as the pits are deepened, grades of haulage roads increase and waste disposal areas are pushed farther away. To counter this problem the producer on the east end of the deposit, Baroid Sales Division, National Lead Company, has started to install a large belt-conveyor system, designed to handle 1300 tons an hour, and the producer on the west end of the deposit, the Magnet Cove Barium Corporation, is planning to begin underground operations.

Both producers resort to flotation concentration methods. Magnet Cove Barium Corporation at Malvern floats barite only.

The National Lead Company at Magnet Cove floats both quartz and barite. The following is a brief description of the latter process.

Flotation Concentration Method

The wet crushing plant operates on a 24-hr basis and consists of a 20 × 30 in. jaw crusher, a ¾ in. vibrating screen, and a set of 42 × 16 in. rolls. Screen undersize goes to a 48 in. × 23 ft screw classifier. The overflow goes to a 20-ft hydroseparator where the coarse sands are separated and pumped to the low-grade grinding circuit. Hydroseparator overflow goes to a 40-ft thickener where the pulp is pumped to the barite fatty-acid flotation circuit. Crushed ore from the screw-classifier underflow is belt conveyed to a six-compartment, 1000-ton mill bin. From this bin the ore is drawn by belt conveyors and delivered to the jig-feed bin. The jigs are Bendelari 3-cell machines. Cup product and tails go to separate dewaterers called "high grade" and "low grade." The high-grade grinding circuit, an 8 ft × 42 in. Hardinge with an 11-ft Dorr bowl classifier, feeds the amine or silica flotation circuit, producing about 65 pct of the plant tonnage and making a product that will assay 81 pct $BaSO_4$, 4.00 specific gravity. The low-grade grinding circuit draws its feed from the jig tails, the sands from the crushing plant, and middlings from the barite-flotation circuit. It consists of an 8 ft × 36 in. Hardinge mill and a 15-ft Dorr bowl classifier and feeds the barite-flotation circuit. Overflow from the high-grade classifier is thickened and then treated in two Bird 36 × 50 in. centrifuges where it is deslimed before flotation. The amine circuit has 5 Fagergren and 6 Denver Sub A cells. In this circuit the gangue minerals are floated away from the barite. The froth is sent to the low-grade circuit. Classifier overflow from the low-grade grinding circuit is thickened and conditioned for fatty-acid flotation. In a 12-cell rougher bank a rougher concentrate is recleaned in three stages to finished product quality. Flotation middlings here are sent to a hydroseparator where the sands are returned to the low-grade grinding circuit. Concentrates from both flotation circuits go to a 50-ft thickener, are filtered, dried in rotary gas-fired driers, and the finished product, containing 94 pct $BaSO_4$, specific gravity 4.30, is hauled in hopper tank trailers 5 miles to the packing plant at Butterfield, the closest railroad point.[a] A 4000-ton storage silo at Magnet Cove and a 3500-ton storage silo at Butterfield provides ample surge capacity. At Butterfield a 4-tube packer fills 100-lb paper bags at the rate of 1 ton a minute and the ground barite is finally ready for carload shipments.

Nevada

At present no ground barite is produced in Nevada, although several producers are conducting mining operations in that state and shipping into California for further processing. Most of the barite activity in Nevada is centered around Battle Mountain, east to Argenta and west to Golconda, although there is a small amount of production farther south from near Tonopah. Several producers hold extensive reserves in northern Nevada, and the Battle Mountain area will become of increasing importance within the next few years. Most of the Nevada production comes from surface operations where the barite occurs as limestone replacement deposits, one of which contains more than one million tons of relatively high-grade ore.

California

In California most of the 1945 production came from a property operated by the Baroid Sales Division, National Lead Company at El Portal, in Mariposa

[a] Since this paper was written a railroad extension from Butterfield to Magnet Cove has eliminated the need for truck haulage of the finished product.

County. This property is the oldest current producer in the state and has for many years been the most important underground barite mine in the United States. It was operated more or less continuously between 1910 and 1927, and from the latter date to the present time, has operated consistently as the largest producer west of the Rocky Mountains. The property has also been noted for the only commercial deposit of witherite in the United States. Although witherite occurs as a minor constituent in most of the barytes, one lens was predominantly witherite. However, it was exhausted several years ago. The barite deposits are about a mile west of El Portal and lie on both sides of the Merced River Canyon. The deposits are lenticular limestone replacements in schists and siliceous limestones. The lenses are almost vertical, varying from 80 to 200 ft long and from 15 to 40 ft in maximum widths. Depths have varied up to 400 ft. While the barite in many of the lenses is high grade, it contains waste inclusions as stringers and horses of siliceous lime. Most of the ore has been developed and mined through adits and raises and until recently no sinking was necessary. Some ore is now being mined from winzes. Original mining methods consisted mostly of quarrying or mill-holing through raises run to the outcrops but present methods are almost exclusively confined to shrinkage stoping with occasional pillars. All ore mined from upper levels is bypassed through transfer raises or old stopes to bunker levels. The mill is on the north side of the river and ore from the south side is conveyed by a 3000-ft semi-automatic double rope tramway to a terminal bin at the mill where the half-ton buckets are automatically tripped. Ore mined from the north-side workings is hand trammed to a bin adjacent to the tramway bin. South-side ore is primary crushed before tramming. North-side ore goes through another jaw crusher beneath the bin before the combined material passes over a vibrating screen in closed circuit with a cone crusher. The crushed material is then jigged. Cup and hutch products go to the grinding circuit, and the waste is stacked. It was formerly used by the Yosemite Valley Railroad for track ballast but that carrier discontinued operations in 1945. Grinding is accomplished in an 8 ft × 36 in. Hardinge mill in circuit with a Dorr bowl classifier and the method is worthy of a few comments. The ball mill is used as a gravity concentrator, a practice which originated at the El Portal plant and has been called "selective grinding." In theory the pulp density of the mill charge is maintained at 3.4 so that the lighter waste floats in the pulp and is discharged unground. The barite being softer, heavier and more friable is readily ground and a separation of ground barite and unground waste is effected by 3/16 in. and 30-mesh trommel screens attached to the discharge end of the mill. The bowl classifier is also in circuit with a 50-mesh trommel screen and all mill discharge coarser than 50-mesh is discarded from the circuit. Plant experience has shown that a better concentration can be effected by selective grinding than by jigs or tables. The ground barite is dewatered in a 30-ft thickener and dried on two rotary steam-film driers before being packed in 100-lb paper bags. Since the railroad is no longer in operation the bags are trucked 70 miles to Merced, the closest railroad point, where they are either loaded into box cars, or continued by truck to oilfield distributing points.

Dry Grinding

High-grade barite, that is, plus 4.25 specific gravity, is usually considered satisfactory without concentrating. The preferred method of grinding, after crushing to minus 1-in., seems to be in roller mills. Recently, one producer has been grinding ore mined and shipped from Nevada in a Los Angeles dry-grinding plant where the barite is crushed and then

ground to 95 pct minus 325-mesh in a Raymond 4-roll, high-side mill. A similar treatment has been used by another producer on ores from Nevada and northern California in a dry-grinding plant at Berkeley.[a]

Arizona

Statistics for 1946 will show some ground-barite production from Arizona where gravity concentrating methods have been tried at a new plant by the Arizona Barite Company near Mesa. It is understood that this method has not been completely satisfactory and that a change to flotation methods is now contemplated.

Reserves

The United States is definitely not a "have not" nation as far as barite is concerned. There are, literally, hundreds of known deposits in the United States, large and small, high and low grade. Producers who are searching for new properties are primarily concerned with comparative economics. How do combined production and transportation costs at any particular property compare with existing costs at operating properties or with probable costs at other properties which can be placed in production on short notice? Quantity, quality, and location are all of primary comparative importance. Because the industry is highly competitive, present producers are actively interested in exploration and examination.

Although present producers are not inclined to be specific in discussing reserves, a reasonable estimate for inferred reserves at properties now in production or being developed in the United States would be at least 20 years at present consumption rates. This figure will be multiplied many times before all possible reserves are exhausted.

[a] Since this paper was written the Baroid Sales Division, National Lead Company, has substantially decreased production at El Portal and is now dry-grinding Nevada ores at a new plant in Merced, Calif.

Heavy Mineral Deposits of the East Coast of Australia

By N. H. Fisher*

(Denver Meeting, October 1947)

Geographical Distribution

The most important known deposits in Australia of what are commonly referred to as the beach-sand minerals are along the most easterly part of the Australian coast, between Southport, 17 miles north of the Queensland-New South Wales border, and Ballina, 50 miles south of the border, and most of the production has come from this area (Fig 1). Smaller deposits are known to occur farther to the south at intervals for several hundred miles, and beaches have been worked at Yamba, 96 miles south and Woolgoolga, 150 miles south of the border and at Swansea, 60 miles north of Sydney. Important deposits have been shown to exist on north Stradbroke Island and others have been located farther to the north, as far as Tin Can Bay at the south end of Frazer Island. (Fig 2).

Table 1 gives the list of operators with the location of their deposits and workings, their approximate maximum monthly capacity expressed in tons of heavy mineral produced by their concentrating plants before separation into individual mineral concentrates, their methods of mining and concentrating the minerals, and the products obtained. This table applies as of June 1948, but all the operators are remodeling or improving their plants and plant practices will be modified accordingly.

In addition, active boring campaigns have been carried out by Alluvial Gold, Ltd., of Sydney, in the Cudgen-Cudgera area, and by Zinc Corporation Ltd., of Melbourne, on Stradbroke Island, and in other places.

Physiography

Submergence of the order of 100 to 200 ft at the close of the Pleistocene period left the pre-existing hills as promontories and the valleys as deep inlets. Wave action has built bay-bars and sandspits in a northerly direction from the headlands, and lakes and marshes have formed between these bars and the initial post-submergence coast line. These lakes and swamps have been or are being filled in by river-borne sediments or wind-blown sand. Beasley[1] has presented evidence that suggests a recent emergence of the coast line of about 10 ft. The evidence for this belief is the presence of a black sand seam ⅓ mile inland and 17 ft above sea level (Fig 3), and although it is not entirely certain that this seam is of beach formation and owes nothing to wind-blown concentration, other evidence supports the suggestion of recent emergence. Such emergence would undoubtedly aid the formation of bars and sandspits and the easterly progress of the beach front, leaving a series of parallel dunes (Fig 4).

This belt of north-south coastal dunes ranges up to ½ mile in width and as many as 15 lines of dune have been counted (Fig 3). The foredune is generally the highest, up to 25 ft high, and there is in many places a flat platform in front of the dune known as the berm. The dunes behind the foredune are lower and less well defined

Published with the permission of the Director, Bureau of Mineral Resources, Geology and Geophysics. Manuscript received at the office of the Institute Oct. 28, 1947. Issued as TP 2455 in Mining Technology, November 1948.

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[1] References are at the end of the paper.

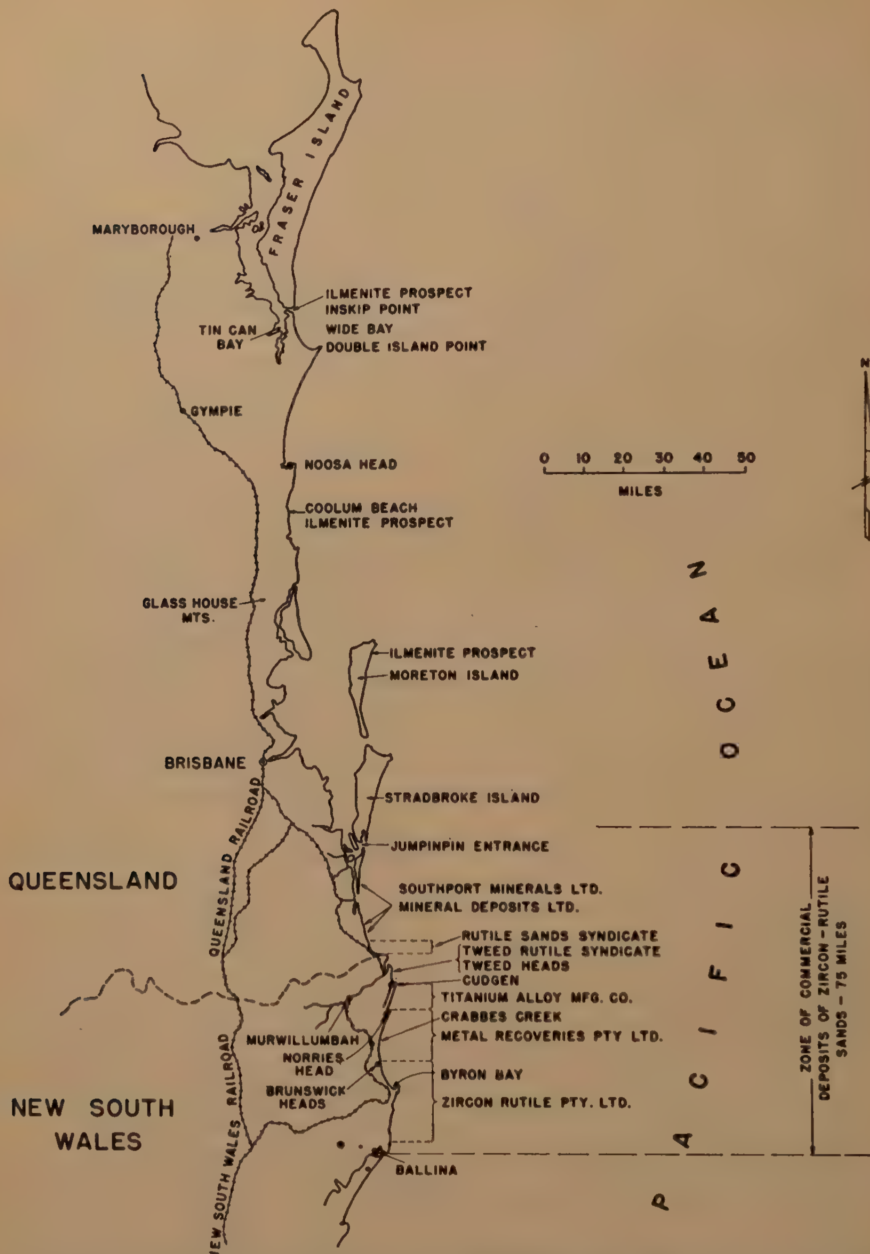

FIG 1—MAP OF A SECTION OF EAST COAST OF AUSTRALIA BETWEEN LATITUDE 24° AND 30° SHOWING LOCATION OF ZIRCON-RUTILE SAND DEPOSITS AND ILMENITE PROSPECTS.

(*Plan by J. L. Gillson*)

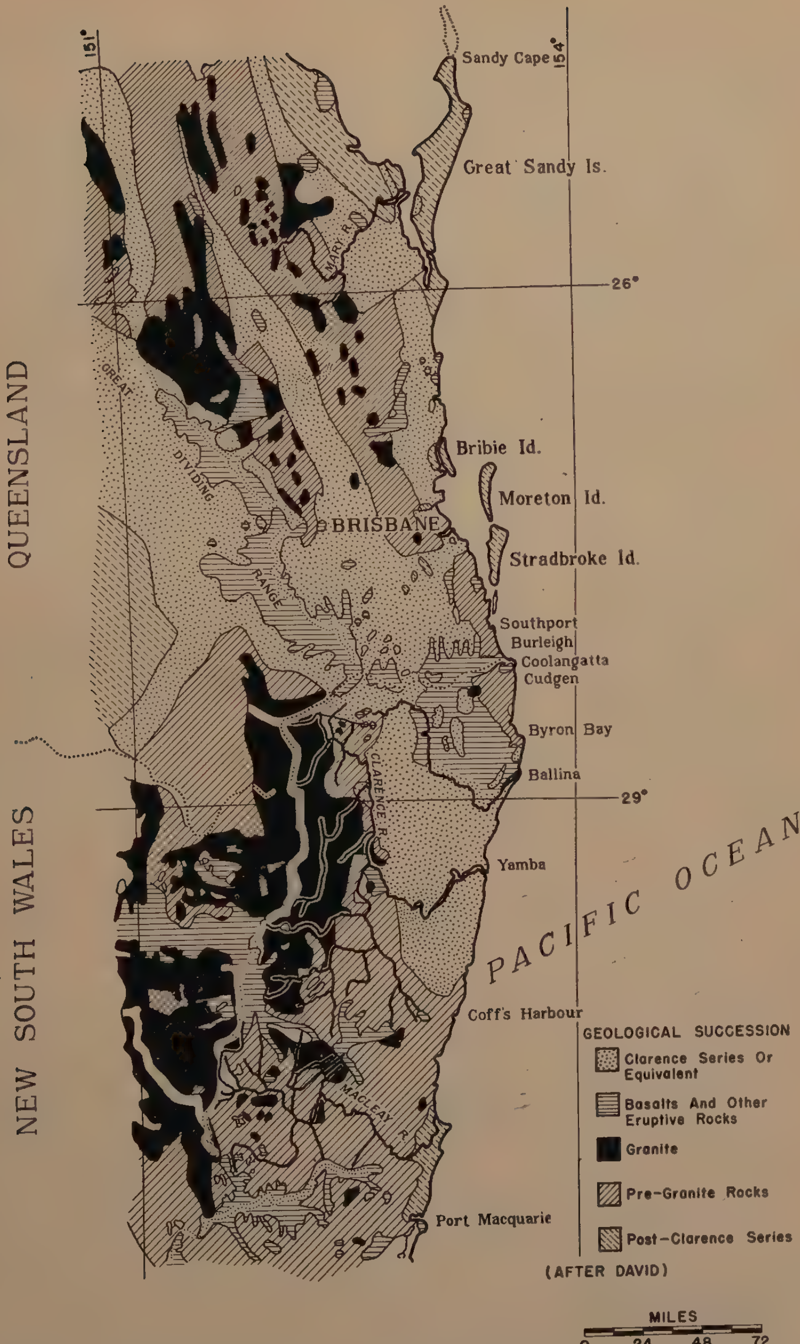

FIG 2—GENERALIZED GEOLOGICAL PLAN OF EAST COAST OF AUSTRALIA FROM PT. MACQUARIE TO SANDY CAPE.

TABLE 1—*Principal Beach Sand Operators*

Operator	Location of Plant 1. and Deposit 2.	Approximate Monthly Through-put, (As Mixed Concentrates) Tons	Method of Mining	Outline of Treatment Method	Products Marketed, Per Cent
Mineral Deposits Syndicate.	1. Southport, Qld., 17 miles north of N.S.W. Border. 2. Broadbeach-Burleigh Area.	600	Stripping overburden by bulldozer, selective hand-loading into motor trucks, replacement of overburden.	Wilfley tables, draining, rotary drier, rotary magnetic separators to eliminate ilmenite, electrostatic separation of zircon-rutile, cleaning by magnetic separator.	Zircon, 85 Rutile, 95 Zircon-rutile, 55:40
Associated Minerals....	1. Southport, Queensland. 2. Southport-Broadbeach Area.	500	Stripping overburden by bulldozer, selective hand-loading into motor trucks, replacement of overburden.	Wilfley and curvilinear tables, rotary drier, electromagnetic and electrostatic separation of zircon and rutile, with further cleaning of each product by both electrostatic and electromagnetic methods.	Zircon, 93 Rutile, 96
Rutile Sands Pty.......	1. Currumbin, 6 miles north of Border. 2. Tugun and Currumbin beaches.	650	Stripping overburden with small bulldozer. Selective hand-loading into motor trucks.	Wilfley tables, draining, rotary drier, electrostatic and electromagnetic separation.	Zircon, 90 Rutile, 96
Tweed Rutile Syndicate.	1. Cudgen Beach, N.S.W., 9 miles south of border. 2. Cudgen and adjacent beach area.	500	Stripping overburden with horse-drawn scoops; loading motor trucks with diesel shovel.	Wilfley tables, conveyor to drier, electrostatic and electromagnetic separating and cleaning.	Zircon, 95 Rutile, 96 Zircon-rutile, 60:40
Titanium Alloy Manufacturing Co. Ltd.	1. Cudgen beach, N.S.W., 10 miles south of border. 2. Cudgen and adjacent beach area.	900	Removal of overburden by power scoops and piling of heavy mineral which is loaded by small drag-line scraper into 2-ft gauge railway trucks drawn by diesel locomotive.	Wilfley tables, draining, rotary drier; 6 electrostatic units; removal of ilmenite by Exolon magnetic separators.	Zircon, 98 Rutile, 96
Metal Recoveries Ltd...	1. Crabbe's Creek and Mooball Siding, N.S.W., 24 miles south of border. 2. Cudgera to New Brighton.	250	Removal of overburden by horse scoops, selective hand loading into lorries.	Wilfley and curvilinear tables at Crabbe's Creek. Concentrates carted to Mooball, dried, passed through Exolon electromagnetic separator and through electrostatic separator.	Zircon, 97 Rutile, 97 Zircon-rutile, variable grade
Zircon Rutile Ltd.......	1. Byron Bay, N.S.W., 34 miles south of border. 2. Seven Mile and Tallow Beach, Byron Bay.	1,000	Overburden removed and heavy mineral stacked and loaded into motor trucks by over-loader; a drag-line loader is also used.	Wilfley tables at beach, concentrates carted to main plant at Byron Bay, zircon taken out by flotation, tailings passed over cleaner curvilinear tables, then through drier and ilmenite and other slightly magnetic minerals removed magnetically; zircon concentrate is dried and cleaned magnetically.	Zircon, 99.5 Rutile, 96
Swansea Minerals[a].....	1. Swansea beach, 60 miles north of Sydney. 2. Swansea.	300	Hand-loading into lorries.	Wilfley tables only.	Mixed concentrate Zircon, 44 Rutile, 10 Ilmenite, 41 Others, 5

[a] Not working at present.

and the most landward dunes are generally broadest. The width of the beaches at low tide is 100 to 200 ft.

movement of the heavy mineral along the beaches, and deposits tend to be ephemeral unless they are anchored at the north end

FIG 3—DIAGRAMATIC SECTION FROM BEACH TO FOOTHILLS OF THE COASTAL RANGE, SOUTH COAST OF QUEENSLAND.
(*After A. W. Beasley.*[1])

by some natural bar. Then the deposit builds southward and may attain several miles in length.

Ocean currents do not appear to play an important part, as the general offshore current along the east coast of Australia flows in a southerly direction with a velocity of 1½ to 2 knots, although this may be reduced, or even reversed in places, near the shore (Halligan[2]).

The whole cycle of processes involved in the concentration of the heavy minerals may be summarized as follows:

1. During normal weather wave action gradually moves sand on to the beach and builds up the beach profile to a comparatively steep angle, approximately 8°.

2. Strong winds, usually from the southeast, blow the lighter quartz sands towards the dunes and leave the heavy minerals concentrated in a thin layer, less than ½ in., on the beach. Repetition of this process may form a series of thin black layers separated by ordinary beach sand.

3. Storm waves as described above concentrate the black sands, remove the quartz and flatten the beach profile to about 4°, leaving a wave-cut cliff at the inshore side of the beach.

Recent prospecting on Stradbroke Island indicates that stage 2 may be of much greater importance locally than was previously realized and that high dunes containing large low-grade deposits of heavy mineral may be built up by strong winds

Manner of Formation of Heavy Mineral Seams

The heavy mineral deposits are formed initially on the beaches, between low-water mark and the highest point reached by storm waves at high tide, by the concentrating action of the surf waves on the heavy mineral content of the beach sands, which in general is considerably less than 1 pct, probably of the order of 0.1 pct. The east coast of Australia is in the belt of the south-east trades and is exposed to continual southeasterly winds. Most of the work of concentrating the heavy minerals is done during storms, particularly during the very violent storms accompanied by fierce gales which occur only once every few years.

Essential conditions for the formation of a deposit apparently include: (1) a stretch of beach running nearly north-south with a high dune at the back and open sea unobstructed by islands or headlands in front; (2) a headland, rock outcrop or other natural bar or a river or creek mouth at the north end of the beach; and (3) an adequate source of heavy minerals in the rock formations of the adjacent areas.

The heavy seas whipped up by the gales strike the beach at an angle (Fig 4) and the surf dashes up the beach and to the right, i.e. in a northerly direction. The returning water has sufficient velocity to carry only the quartz sand with it, leaving a residual concentration of the heavy minerals. The result of this process is a continual northerly

continually robbing the beaches of their wave-borne heavy mineral content.

From the manner of their formation the individual beach deposits tend to be lenticular in cross section and their distribution is erratic unless they are stabilized by a protecting bar at the north end of the beach as indicated above. They extend from somewhere between high and low-water mark to a point reached by the strongest storm waves some distance above high-water mark. The deposits become thinner towards the south and may split into two or more seams. In cross section they feather out gradually on the seaward side as the almost flat base converges with the slope of the beach surface. At the landward side they terminate rather abruptly, immediately after attaining maximum thickness, which may be as much as 5 ft but is usually only 1 or 2 ft in individual seams (Fig 5–8). They are usually not more than 50 ft in width, except where the deposit has grown gradually seawards with a slow easterly progression of the beach, and extends continuously through more than one line of dunes.

The building up of a large deposit is a process which takes a considerable number of years although it is not a slow process by geological standards of time. Statements have been made that beach deposits that have been worked out are regenerated by severe storms but this happens only where

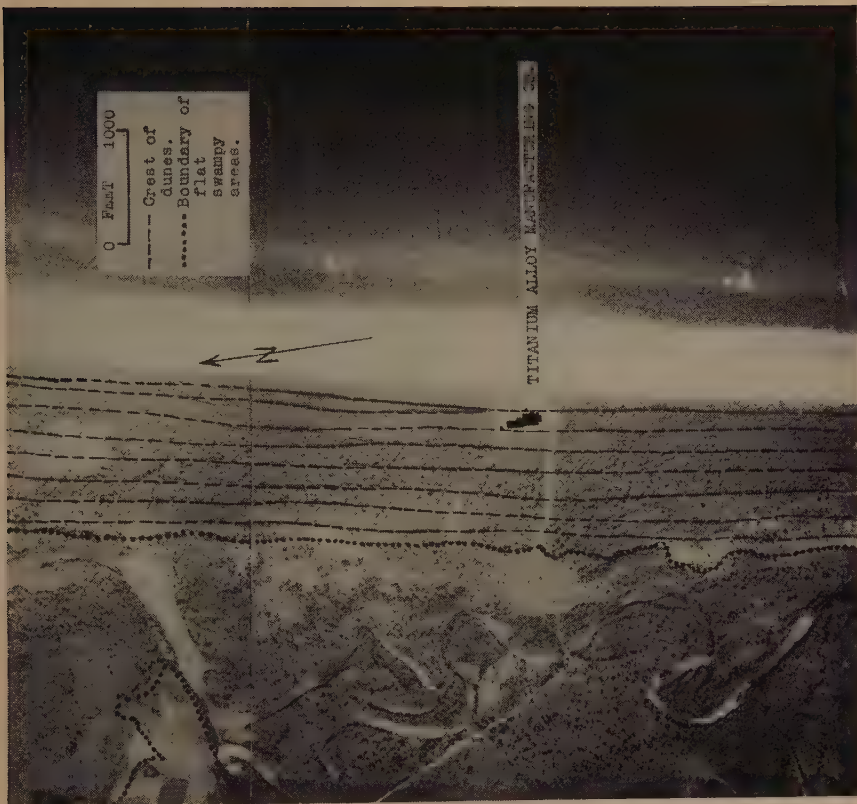

Fig 4—Aerial photograph of Cudgen Beach.

part of the northerly section of the beach has been worked and storms have moved heavy mineral northwards from the unworked section to the "vacant" space made available by the workings, or where only high-grade concentrate has been mined from a deposit, leaving sufficient black sand in overburden or rejected narrow seams to be reconcentrated into one workable seam. No case has been observed of regeneration of a deposit that has been thoroughly worked.

The position of the water table relative to the base of the deposit is important to the operator and in most cases, particularly where a line of dunes is flanked on the landward side by low swampy areas, the water table obviously corresponds to sea level and approximately to the base of the deposit. Where rising ground occurs behind the deposits, the water table may show a corresponding rise and seams may be found below ground-water level. However, no concentrations of heavy minerals have been found below sea level (at low tide).

Origin of the Heavy Minerals

The black-sand deposits, as explained above, are merely the result of the concentration of the heavy mineral content, usually about 0.1 pct, of the beach sands, and are likely to occur anywhere that a suitable combination exists of coast orientation relative to prevailing winds and currents.

Thus deposits are known to exist at intervals all along that part of the east coast of Australia, south of the Great Barrier Reef, that is exposed to the southeast trades. However, the relative abundance and richness of the deposits in the area from Ballina to Stradbroke Island must be related to the existence of a comparatively prolific source of zircon, rutile and ilmenite in the mainland rocks of the area. The main immediate source usually has been assumed to be the fresh-water sandstones of the Clarence series of Triassic-Jurassic age, with some contribution of ilmenite from the basalts of the Ballina-Point Danger area. The original source of most of the zircon, rutile, monazite, and part of the ilmenite is undoubtedly the extensive Permian granite masses of the New England area (Fig 2), which extend from just north of the Queensland border south for 250 miles. After erosion of the granite these minerals have eventually found their way to the beaches, either via the Clarence sandstones or more directly by way of the present major streams of the north coast area, which rise in, or adjacent to the granite. The principal stream is the Clarence River which enters the sea near Yamba, 96 miles south of the border and this river, together with the Richmond which comes out at Ballina, (or their ancestral streams) is considered to have been the main avenue of delivery of heavy minerals to the ocean sands.

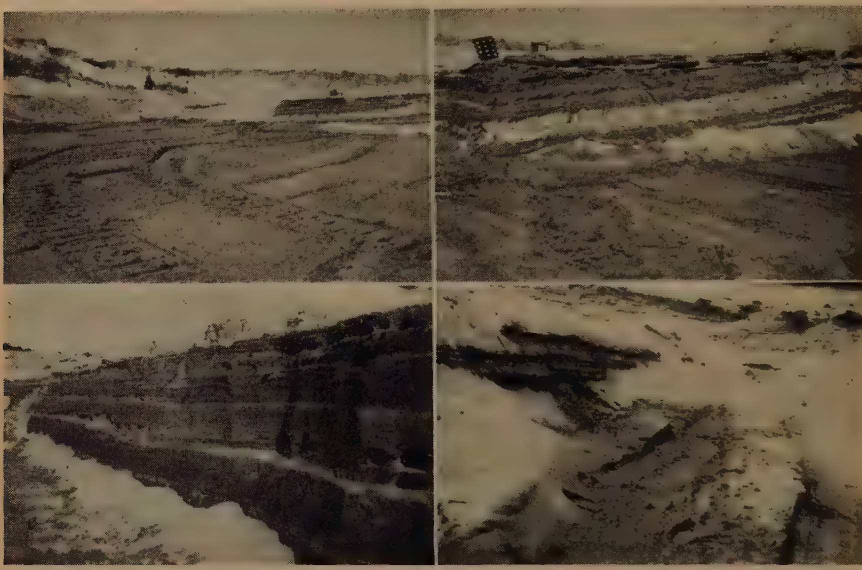

Fig 5—General view of workings south of Broadbeach behind the fore dune 250 ft from beach.
Fig 6—Closer view of workings south of Broadbeach.
Fig 7—Black sand seam 4 ft-3 in. thick south of Broadbeach behind the fore dune.
Fig 8—Black sand seam extending into fore dune. Combined thickness 3 ft-6 in.

Beasley[3] found that a sample of the Clarence sandstones from near Byron Bay gave a good yield of heavy mineral of the approximate composition: rutile, 50 pct; zircon, 25; ilmenite, 15; others, 10. A sample of graywacke obtained inland from Southport gave a small yield of minerals consisting of 85 pct zircon, 10 pct ilmenite and 5 pct other minerals. (In the Sydney area samples from beach deposits taken between Port Kembla, 50 miles south and Swansea, 60 miles north of Sydney, gave a

zircon-rutile-ilmenite ratio of approximately 44:14:42, and Whitworth[4] has recorded heavy mineral contents of the Triassic sandstone about Sydney with a zircon-rutile-ilmenite ratio of 40:15:45. Samples of concentrates from Swansea gave a zircon-rutile-ilmenite ratio of 46:11:43.)

Composition

The heavy mineral content of the deposits that have been worked ranges from 20 to 80 pct, but probably averages, in the feed to the concentrating tables, 40 to 50 pct by volume. Most of the operators practice selective mining in addition to removal of overburden, which may be as much as 20 ft thick (Fig 5), and where practicable discard barren or low-grade seams.

The mineral composition of the heavy mineral concentrates obtained ranges in general from 44 to 70 pct zircon and 15 to 30 pct each rutile and ilmenite.

TABLE 2—*Approximate Average Composition of Heavy Mineral Concentrates*

Locality	Zircon	Rutile	Ilmenite	Other Minerals
Collaroy	40	15	43	2
Swansea	44	11	41	4
Woolgoolga	28	34	25	3
Wooli	70	10	18	2
Yamba	70	13	15	2
Ballina	62	17	16	5
Byron Bay	54	26	18	2
New Brighton	50	25	23	2
Cudgen	48	27	23	2
Currumbin	51	25	22	2
Palm Beach	49	25	23	3
Burleigh	50	22	25	3
Broad Beach	45	28	25	2
South End South Stradbroke Island	43	27	26	4
South Stradbroke Island	30	24	44	2
North Stradbroke Island[a]	26	16	56	2
South Moreton Island	23	20	55	2
North Moreton Island	21	15	60	4
Bribie Island	18	19	60	3
Caloundra	21	18	58	3
Noosa	19	14	63	4

[a] More recent grain counts of beach sand samples averaged: zircon, 30 pct; rutile, 34.5; ilmenite, 29.5; other minerals, 6 (Connah[7]).

The concentrates considered as a zircon-rutile-ilmenite product are remarkably clean. The sum of the other heavy minerals seldom exceeds 5 pct and is usually about 2 or 3 pct. The most abundant of the minor constituents are garnet, monazite, tourmaline and cassiterite, but the amount present of any one of these rarely exceeds 1 pct, although local concentrations of garnet have been noted. Other minerals recorded include spinel, leucoxene, epidote, chromite, pyroxenes, andalusite and staurolite. Table 2 gives the approximate average composition of the concentrates from representative beaches, obtained partly by grain counts carried out by A. W. Beasley,[3] by H. F. Whitworth[4] of the New South Wales Department of Mines and by the Bureau of Mineral Resources, partly from production records or other information supplied by the operating companies.

The proportions of zircon, rutile and ilmenite in the concentrates have been plotted relative to distance north of the mouth of the Clarence River (Fig 9). The graph illustrates rather strikingly a gradual decrease in the proportion of zircon and an increase in rutile and ilmenite as far as South Stradbroke Island, then an abrupt decrease in zircon and increase in ilmenite, while rutile decreases slightly. North of Stradbroke Island the composition remains fairly constant. The writer's interpretation of these variations is as follows. What may be termed the normal proportion of zircon, rutile and ilmenite in natural concentrates derived from the rocks of southeast Queensland is approximately that found north of Stradbroke Island, viz. 22:16:62. The principal source of zircon is the streams draining the area occupied by the Clarence sandstones and the New England granite, of which the main one is the Clarence River. This area has also contributed a relative enrichment of rutile. As the minerals were drifted northwards along the coast the variation in composition as far as South Stradbroke Island was only such as might be accounted for by minor local additions, particularly of ilmenite from the basalt areas, and differences in distribution due to

the relative specific gravities of the various minerals. North Stradbroke Island, which is 24 miles long, 7 miles wide at the northern end, with an area of approximately 107 square miles, is built up almost entirely of sand dunes, rock outcrops being confined to a very small area at the northern end.

The sand dunes on Stradbroke reach a maximum height of 719 ft and those on Moreton Island to the north, which is similar in every way, have a maximum height of 919 ft.

These islands, and particularly Stradbroke, have acted as a vast sand trap and have held the "excess" minerals drifting along the coast from the Clarence area. Hence north of Stradbroke the composition of the concentrates returns to the "normal" for the southeast Queensland area and stays constant, as the influence of the Clarence series and the New England granite has failed to penetrate north of Stradbroke Island.

A point of special interest with regard to the ilmenite of the east coast beaches is its chromium content, which ranges in assays made of the clean ilmenite concentrates up to 5.2 pct Cr_2O_3. Information now available indicates that the chromium is present partly as chromite grains, which follow the

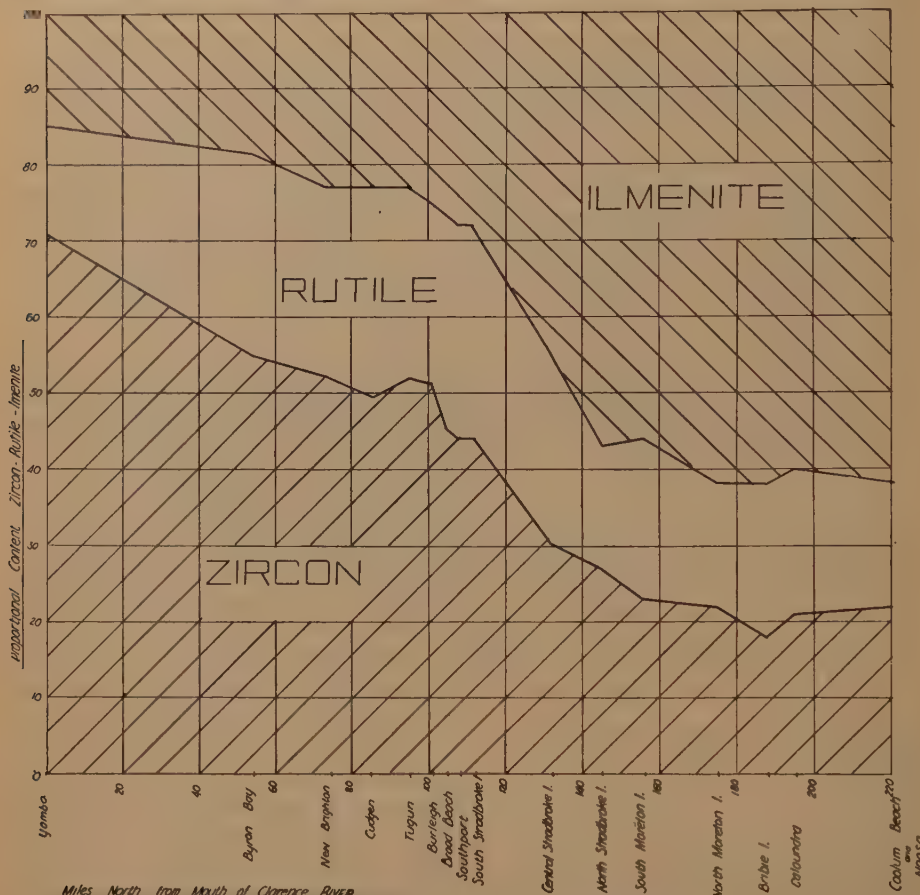

FIG 9—GRAPH SHOWING VARIATION IN ZIRCON, RUTILE AND ILMENITE IN BEACH SAND CONCENTRATES, WITH DISTANCE NORTH OF MOUTH OF CLARENCE RIVER.

ilmenite in the separation processes employed, and partly in combination within the ilmenite. Further work is required to determine the distribution of the chromium more exactly. The composition of the pure ilmenite apparently corresponds to the standard formula for ilmenite ($FeO \cdot TiO_2$) with a theoretical TiO_2 content of 52.7 pct.

Reserves

At the present time it is not possible to give any concise figures for reserves of the beach-sand minerals. Boring campaigns are being carried out and much further work will be done in connection with the Commonwealth Government's campaign to determine accurately the monazite reserves. Although the beaches on which deposits occur are distributed along more than 100 miles of coast, the reserves contained in the actual beach deposits are obviously limited and probably would not be sufficient to maintain production at the present rate (21,576 tons of zircon and 13,194 tons of rutile in 1947) for more than 10 or possibly 20 years. However, very much larger reserves are contained in the deposits belonging to earlier sand dunes behind the present beaches, particularly in the Byron Bay, New Brighton-Cudgera, Cudgen and Currumbin-Southport areas and on North Stradbroke Island. Recent boring in the old dunes behind the beach in the Cudgen area (Fig 4) has indicated large reserves. Boring has shown that many deposits of workable grade exist in the old dune lines on the Queensland side of the border but the quantities available in this section are drastically reduced by the fact that as this is Queensland's premier pleasure resort, the land has mostly been taken up for residential purposes and much of it has already been built upon. The most significant results obtained recently with regard to reserves however are those obtained by Zinc Corporation Ltd. on Stradbroke Island. West of the coastal dunes along the east coast of Stradbroke Island is a swamp up to ½ mile in width and behind this again are old dunes 200 ft or more in height. Boring with a hand-operated posthole auger to a maximum depth of 22 ft on these dunes disclosed the presence at least to that depth of up to 10 pct of heavy minerals, obviously of wind-blown origin, containing approximately 30 pct zircon, 25 pct rutile and 45 pct ilmenite.[a] Although the average heavy mineral content indicated by these bores is probably not more than 3 pct, the enormous quantities of sand that appear to be present give promise of very large reserves of zircon, rutile and ilmenite. Deep boring with a power-driven plant has confirmed the results of the shallow boring and at present further work is being undertaken to test thoroughly the mineral content of the high dunes. The effects of the discovery of these large quantities of zircon and rutile upon the economy of the zircon-rutile industry, both with regard to operators and to consumers, has yet to be determined and is at present being investigated.

Acknowledgments

In addition to the papers listed in the References the writer has had access to many unpublished reports prepared by officers of State and Commonwealth Government Departments and private companies, and has closely followed the progress of the beach-sand industry since 1942. I would like to record my appreciation of the ready co-operation and friendliness of the operators who have always been ready to provide information and discuss problems of heavy mineral occurrence. In particular I wish to thank Mr. A. W. Beasley, who spent two years, 1945 and 1946, carrying out research work on the beach sands of Southern Queensland, for allowing me the use of his manuscript in advance of publication and for providing the photographs which constitute Fig 5 to 8 of this paper.

[a] Private communication.

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The Occurrence and Mining of Solid Bitumens in Western Argentina

By Howard A. Meyerhoff,* Member AIME

(New York Meeting, February 1948)

In western Argentina, in the Province of Mendoza and the Territory of Neuquén, there is a series of solid bitumen deposits which are claimed to be the most extensive in the world. In a linear belt 500 km (311 miles) long, roughly paralleling the axis of the Andes but occupying a variety of positions with respect to the orogenic structures, more than 100 outcrops have been reported. Approximately 70 of these have been prospected, and some mining has been planned or attempted for about half this number. Limited tonnage, high sulphur content, inaccessibility, labor shortages, and disastrous fires have taken a heavy toll among these operations, and in 1947, only 4 deposits were being actively mined, but 8 more were being explored and developed with a view to placing them in production. Inasmuch as these deposits are currently yielding a substantial fraction of Argentina's limited domestic supply of solid fuels, a brief account of the deposits and of the small industry based upon them will provide a commentary on Argentina's fuel problem.

Character of Solid Bitumens

The solid bitumens of Mendoza and Neuquén are petroleum derivatives, as is indicated by the following ultimate analysis:

Constituents	Per Cent
Hydrogen	6.8
Carbon	82.4
Nitrogen	1.6
Oxygen	2.7
Sulphur	2.2
Ash	4.3
	100.0

Manuscript received at the office of the Institute Feb. 24, 1948; revision, Aug. 16, 1948. Issued as TP 2480 in Mining Technology, November 1948.

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In physical appearance they bear a close resemblance to coal and are commonly called *carbón*. Their specific gravity has an extreme range of 1.10 to 1.777 in 36 samples weighed, but the average is between 1.14 and 1.24. The material is black, lustrous to dull, with blocky to sub-conchoidal fracture. Though brittle, the purest material is softer than anthracite and exhibits slightly more cohesion. This bituminous material is generally known as *asfaltita* or *asphaltite*, but definitive chemical studies indicate that the term *asphaltite* or *grahamite* can be applied correctly only to the exceptionally pure material of the Auca Mahuida district in Neuquén. Some other name should be used for the chemically variable bitumens found in all other districts, and the designation "asphaltitic pyrobitumen" has been proposed by Carlos A. S. Piscione.[1]

At the present time these bitumens are finding favor as solid fuel and as a gas-producing medium. They yield heat values of 8650 to 9750 cal,[2] or 14,500 to 16,750 Btu per pound, and in consequence of their excellent heating qualities and the scarcity of imported coal, they are marketed as far away as Buenos Aires at a price of 90 to 95 pesos, or approximately $22 to 23, per metric ton. In burning, the material swells and decomposes into solid, fluid, and gaseous fractions which burn violently though nonexplosively. Its combustibility has been responsible for disasters and virtually complete destruction at Mina La Esperanza and Mina Santa Marta, and for minor mishaps elsewhere, as at Mina Escondida in August 1947.

[1] References are at the end of the paper.

The chemically pure grahamite in the deposits of the Auca Mahuida district in Neuquén is being utilized in part for its physical and chemical properties, and it commands a price of 300 pesos ($73) per ton. Little attention has been paid to the chemistry of the bitumens in the other deposits, although it seems probable that they, too, could command a higher price in the chemical market than in the fuel market. Many fuel or proximate analyses have been made, yielding variable results, which may be summarized as in Table 1.

TABLE 1—*Representative Analyses of Bitumens*[1]

Mine	H_2O	Volatiles	Fixed Carbon	Ash
La Valenciana......	0.34	31.29	68.99	0.38
Minacar...........	0.73	44.82	54.02	0.43
San Eduardo.......	0.15	29.84	68.45	0.55
Escondida..........	0.57	61.30	39.36	0.76

Coking tests reflect the chemical variability which is evident in the proximate analyses. In those samples best adapted to coking, there have been yields of 70 pct coke, 20 pct gas (approximately 12,000 to 12,500 cu ft per ton); 5 pct tar (10 gal); 2 pct light oil; 0.2 pct ammonia; and about 1.0 to 2.0 pct of residual liquor. Analyses of the gas are few but indicate the presence of 40 to 60 pct CH_4; 20 to 45 pct H_2; 10 pct illuminants; and up to 4 pct CO and CO_2. In most tests the coke produced has a low density and lacks the characteristics requisite for metallurgical coke; and the tar and light oil byproducts have a high content of the aromatic compounds, including napthalene ($C_{10}H_{18}$) and benzene (C_6H_6). In general, bitumens relatively high in fixed carbon, like those from La Valenciana and San Eduardo, give the best coke yields and are comparable to a bituminous coal with a high volatile content. Those bitumens in which the volatile matter exceeds 40 pct, like the sample from Minacar, fuse to such an extent that it is impractical to coke them, but they yield larger quantities of gas and oil distillates. The grahamite from Escondida, in which the volatile matter exceeds 55 pct, behaves essentially like an oil of mixed base. Limited as the detailed chemical information is, it suggests that use of the bitumens solely for their fuel value entails considerable economic waste, and that greater values could be obtained if they were utilized in the chemical industries, where coke for domestic or light industrial heating would also be obtained as a by-product.

LOCATION OF DEPOSITS

The northernmost group of prospects is near the headwaters of Río Diamante at latitude 34°, 150 airline km (93 miles) from the railhead of the Buenos Aires and Pacific Railroad at San Rafael (Fig 1). They lie well within the Andes, and one prospect, El Condor, is inaccessibly situated 500 ft below the continental divide at an elevation of 12,000 ft. Active prospecting under the direction of the Board of Solid Fuels is proceeding on two deposits in this area, but unless large tonnages of marketable grade are found, the cost of road construction will be prohibitive to a commercial operation.

The recent extension of the State Railway (Ferrocarriles del Estado) from San Rafael southwestward 175 km (109 miles) to the town of Malargüe at the eastern edge of the Andes has brought rail transportation within easy trucking distance of three districts within the foothills. There is, however, only one mine—La Valenciana, 34 km (21 miles) west of Malargüe—which has known commercial value and a production history. Approximately 100 km (62 miles) farther south lies another group of deposits, among which is the largest producer of asphaltite in the country—Minacar—together with two properties of uncertain value that have been explored and partially developed. A truck haul of 100 km from Minacar will soon be cut to 30 km, for the State Railway is being extended from

Malargüe to Barda Blanca on Río Grande, where the road from Minacar joins Federal Highway No. 40. The construction of this newest railway link and the consequent and Auca Mahuida, 150 to 300 km (93 to 185 miles) from the same line at the city of Neuquén. The network of public roads in the vicinity of Chos Malal is the best

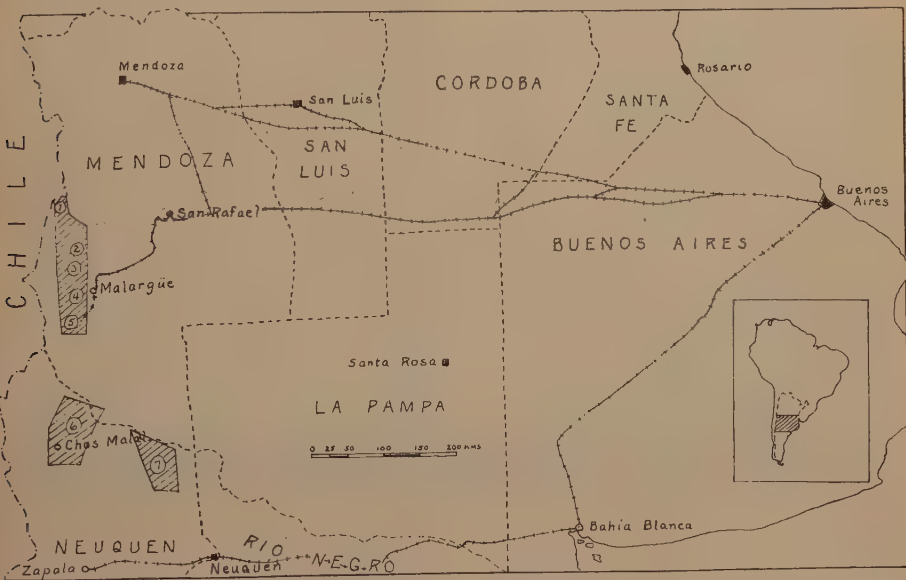

FIG 1—CENTRAL ARGENTINA.

The full width of the country separates the asphaltite deposits from their principal market. The principal asphaltite districts include: (1) Río Diamante, (2) Río Atuel, (3) Río Salado, (4) Río Malargüe, (5) Río Grande or Pala Mahuida, (6) Chos Malal or Pum Mahuida, (7) Auca Mahuida.

reduction in transportation costs may encourage exploitation of several small deposits within easy reach of the railroad. Only one asphaltite district, in the Sierra de Reyes near the Mendoza-Neuquén boundary, will fail to benefit. This district includes but one partially developed and unpromising prospect known as Mina Santa Isabel, which lies high above Río Colorado in such rough country that, even though the new railway will halve the road mileage to a shipping point, it will not alter the local inaccessibility or compensate for the complete absence of water.

In the Territory of Neuquén most of the deposits are located in two districts: Chos Malal, which lies nearly 200 km (124 miles) north of the terminus of the Southern Railway (Ferrocarril del Sud) at Zapala; encountered anywhere south of the city of Mendoza, and local mine managements have but modest construction costs to tie in with the public road system. In the Auca Mahuida district, on the other hand, Mina Escondida, the largest operation in the area, is located 90 km (56 miles) north of the public highway which joins Neuquén and Añelo, on a fair road which is privately maintained; but at Mina La Fortuna, which lies 145 km (90 miles) farther northwest, exploratory work was abandoned late in 1946, because of inaccessibility, lack of water supply, and scarcity of workmen who were willing to live so far out of the world.

Transportation charges constitute the major item in the cost per ton of asphaltite delivered, regardless of the point of consumption. At Mina La Valenciana, for

example, mine costs total 19 pesos per ton, whereas transportation charges to San Rafael, including a 35-km (22 miles) truck haul to Malargüe and a 180-km (112 miles) rail haul from that town, total 29.57 pesos. From Minacar delivery to San Rafael costs 37.57 pesos. Shipment to Buenos Aires, on the other hand, is inexpensive, because the railroad rebates 16 pesos per ton on the first 1000 km (621 miles) of any haul in excess of 1000 km. The rail distance to Buenos Aires is 1161 km (721 miles), and the net cost of delivery f.o.b. Buenos Aires from Malargüe under this arrangement is 21.19 pesos. These costs rise in Neuquén, for both road and rail distances are greater.

Exploration and Exploitation as Geologic Problems

The solid bitumens appear to be residues of oil accumulations that have been fractionated and emplaced by igneous intrusion. All of the deposits except one are localized in the Mesozoic sediments which make up the Andean foothill structure from Río Diamante on the north to Río Agrio and Río Neuquén on the south. The single exception is a small prospect west of Malargüe known as Mina El Toki, in which thin seams or lenses of asphaltite are intercalated with coarsely bedded pyroclastics that dip 25°. The pyroclastic materials appear to be much younger than the associated late Jurassic-Lower Cretaceous sediments, from which the bitumen was obviously derived. The bitumen deposits likewise are restricted to the marine Mesozoic formations ranging in age from Late Jurassic to Middle Cretaceous, although individual veins locally penetrate continental horizons within the section.

Under the mode of origin postulated, oil might be expected in the same stratigraphic series, and this is the case. Oil pools have been found in the milder structures of the Andean piedmont in Mendoza and in the Pampa east of the mountains in Neuquén, where Plaza Huincul provides Argentina with its largest producing pool outside the Territories of Chubut and Santa Cruz. Oil pools have been found, however, only in localities distant from the larger Tertiary intrusives, but formation and migration of petroleum have by no means stopped in the asphaltite districts. Oil-filled vugs were found in the Upper Jurassic section along Río Poti Malal south of Río Grande, about 20 km (12 miles) from Mina Mercedes; a seep was observed on Arroyo Chico, 5 km (3 miles) above its junction with Río Grande and less than 10 km (6 miles) from Minacar;* and minute oil seeps were noted in surface excavations at Mina La Valenciana. Among those who have studied the asphaltite deposits, there is general agreement as to their derivation from oil, but some disagreement as to the manner, method, and date of derivation.[3,4,5] It may be noted that the deposits are clustered subsymmetrically around intrusives (Fig 2), mostly andesitic in composition and Middle to Late Tertiary in age; the volume of solid bituminous material associated with the several intrusives, though variable within wide limits, is sufficiently large to have required the presence of oil pools as source materials at the time of intrusion; emplacement of the asphaltite was not simultaneous throughout the zone, but occurred at dates ranging from Middle to Late Tertiary, and possibly as late as Pleistocene; emplacement occurred in receptive structures, taking place rapidly, with the generation of moderate to high hydraulic pressures.

With three noteworthy exceptions, the solid bitumens in the Province of Mendoza occupy orogenic structures in the folded sedimentary series, exhibiting local conformance with the enclosing strata. Detailed study, however, reveals that the

* Near the junction of these two streams the MAPYSCA Oil Co. has drilled 560 m (1837 ft) on an anticlinal structure, but drilling has been at least temporarily suspended. The proximity of dikes and larger intrusive masses, as well as capping flows, is a dubious asset for a drilling site.

structural relationship is rarely simple: In Minacar approximately 500,000 tons of the solid bitumen were forced into the crest of a small, closed anticline, with localized stoping and lit-par-lit injection at the contacts, especially along the roof. At Mina La Valenciana the asphaltite lies in a shallow trough or syncline (Fig 3). The main ore body conforms with the gently dipping sediments on the roof, but the floor rocks are flexed and faulted; and in places broken and displaced strata form horses within the seam (Fig 4). Maximum accumulation of bitumen occurred along a normal fault, which caused 5 to 6 ft of displacement in the floor without deformation of the roof. This fault parallels the strike and continues for several hundred meters, providing a channel-shaped deposit of considerable value. Beyond the fault, however, the deposit thins, and drifting has not yet exposed a comparable structure which might prolong the life of the mine. At Mina Mercedes the bitumen fills a flexure in a fold. Here, again, the roof, though arched, is unbroken; while immediately beneath it, emplacement generated sufficient pressure to break and displace the sediments and to provide an unusual exhibit of incipient stoping.

Fig 2—Pala Mahuida, a Tertiary andesitic intrusive (background) in Jurassic marine sediments (middle ground).

The intrusive is elongated parallel to the north-south strike and is subsymmetrically situated with respect to the asphaltite deposits of the Río Grande district, Mendoza.

The relation between the Mendoza deposits and the fold structures in the enclosing sediments supports the view that injection of asphaltite was synchronous with the final phases of orogenesis, and most of the andesitic intrusives to which injection is referable appear to have entered the rocks at the same time. Where oil was present, fluid bitumen was driven into any and every available incompetent structure, weak enough to accommodate new material; hence the form and distribution of the asphaltite deposits present serious and, perhaps, insoluble problems in exploitation and mining. Except that the bitumens have come to rest at local points of minimum pressure, and that they characteristically lie within short to moderate distances of andesitic intrusives, they conform with no other principles of accumulation or concentration. Individual deposits were noted in anticlines, synclines,

terrace-like flexures on the limbs of folds, monoclines—in short, in every conceivable type of fold structure. There were, moreover, varying degrees of receptivity among

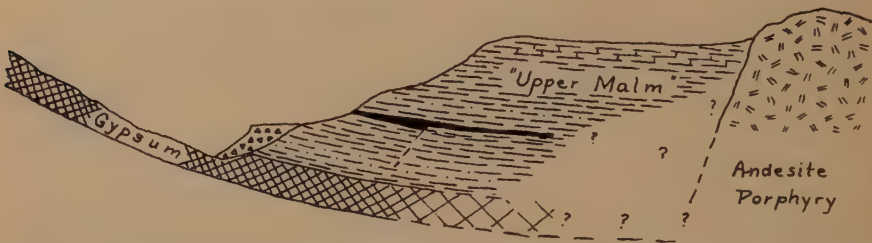

FIG 3—GENERALIZED CROSS SECTION AT MINA LA VALENCIANA. LENGTH OF SECTION APPROXIMATELY 2 KM; VERTICAL EXAGGERATION, 5 TIMES

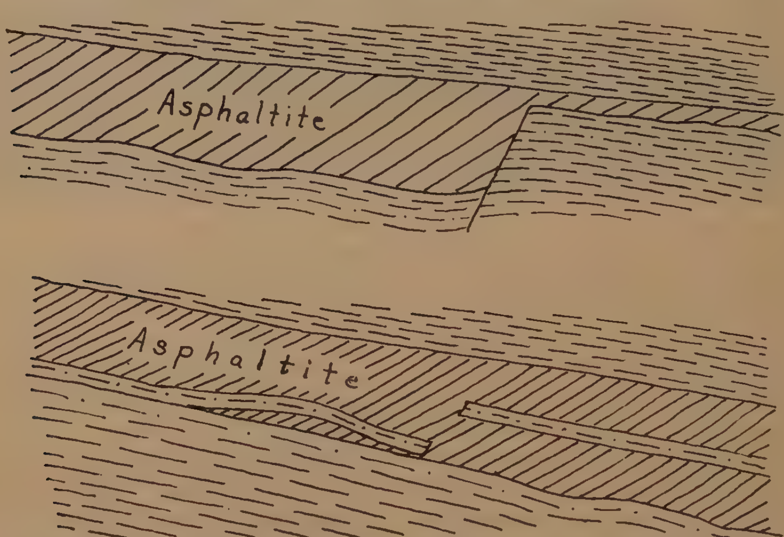

FIG 4—STRUCTURAL DETAIL WITHIN LA VALENCIANA MINE. MAXIMUM THICKNESS OF ASPHALTITE SEAM, 7 FT.

the structures utilized, as well as varying amounts of bitumen forced into them.

In consequence of this heterogeneous set of circumstances, it is impossible to predict where additional deposits may be found, or how much asphaltite can be recovered from known deposits, until they are extensively explored and developed. It is not surprising, therefore, that a mine management, like the one at Mina La Valenciana which urgently needs new reserves to stay in business, can not turn to any tested technique of discovery to guide exploratory drilling, even though commercial reserves may lie within easy reach of mining operations. At the time Mina La Valenciana was examined, 130,000 tons of asphaltite had been extracted; 20,000 more were definitely in sight; with luck another 50,000 tons might be uncovered; but without a major new discovery the operation will have a short life. The most successful operation in the Province, Minacar, is working on a finite deposit of 500,000 tons, 60 pct of which has been extracted. All known deposits in the immediate vicinity have been explored without uncovering reserves for future use, but no economical or

efficient technique is available for subsurface exploration which might disclose new deposits near by or at depths shallow enough to work. In all Mendoza there are only two other prospects which offer definite promise of equaling these two properties. Both are being actively explored by the Board of Solid Fuels; and, unless commercial values are established in one or both of them, asphaltite mining may come to an untimely end in the province. It is unlikely that the accidents of erosion have exposed all the workable deposits, but the vagaries of structure and quantity make the future of the industry precarious.

In the Territory of Neuquén, practically all of the major deposits occur as veins, which cut the sedimentary structures (Fig 5). This relationship holds in the Pum Mahuida or Chos Malal district, where the Mesozoic section has suffered moderate orogenic deformation, and in the Auca Mahuida district, which lies east of the Andean front in an area of flat-lying Middle Cretaceous strata. In the latter district intrusives cut and locally deform the sediments. The asphaltite fills vertical fissures along which there has been minor differential displacement, but the fracture systems appear to be of the radial and annular, or tangential, types characteristic of volcanic centers. Thus far, asphaltite veins have been found only on the eastern and northern flanks of the intruded area.

The numerous bitumen deposits of the Chos Malal district are related to the volcanic mass known as Pum Mahuida, where renewed vulcanism in Quaternary time has rebuilt the old center into the spectacular peak called Volcán Tromen. Here prevolcanic folding exerted some influence on the fracturing induced by vulcanism, and the subsymmetrical location of the intrusion with respect to the oil reservoir was responsible for the injection of asphaltite into vein systems in nearly every compass direction around the Tertiary andesitic mass except the northwest. Maximum concentration of bitumen-filled veins with commercial possibilities occurs to the east and south, and has provided the incentive for five mining operations and a great deal

FIG 5—PARTIALLY MINED OUT VEIN OF ASPHALTITE AT MINA CURA-CO, NEUQUÉN.

of prospecting. In 1947, Mina San Eduardo was the only producer. Two excellent properties were completely wrecked by separate fires in 1943 and 1944, and no attempt has been made to reopen them. Another mine (La Riqueza) was opened, following systematic exploration, careful determination of reserves, and efficient development; but it was forced to suspend operations when consumer resistence developed to the high sulphur content of the asphaltite. Mina Cura-Co is being re-explored and developed, and here, too, the high sulphur content, caused by contamination from near-by gypsum deposits, may give its owners the same marketing problem as confronted the La Riqueza management.

The veins of Neuquén are easy to find, explore, and develop. They are generally straight, but they vary in length from a few hundred meters up to 6000 meters. All of the important ones are vertical—a fea-

ture which simplifies mining and makes it possible to determine reserves with a high degree of accuracy. The veins are not regular but, in pinching and swelling, they give varying widths to the bitumen filling, and locally large pipelike masses have been formed at the intersections of two or more asphaltite-bearing veins. Unfortunately deposits of the vein type acquire depth rapidly, and operating costs and problems mount in direct ratio to depth.

Fig 6—Minacar, Río Grande district, Mendoza.

Mining Operations

Argentina's asphaltite production is based upon a small number of operations, each one of which is in itself small. The largest production (3500 to 5000 tons a month) comes from Minacar in the Río Grande or Pala Mahuida district. It is an efficient mine, operated by a businesslike management that is utilizing sound technical advice in exploratory and extractive operations. The modest monthly output is geared to the small reserves and to the cramped form of the deposit. It was opened in 1941, and has approximately five more years of life at the current rate of production, unless new reserves are encountered at depth. Mina La Valenciana was not seen under ideal conditions. A new and inexperienced management seemed ill-equipped financially and otherwise to recover economically the known tonnage still underground; and without the discovery of new reserves, the operation will soon terminate. At Minas Río Salado and Mallin Largo the ore bodies had been probed rather than developed, and no systematic plan of efficient extraction had been evolved at the time these properties were visited. In the entire Province of Mendoza, the irregularities of shape and size which the individual deposits exhibit encourage amateur improvisation in the mining methods employed, and there is an indifference to efficiency and safety that is difficult to describe and impossible to comprehend.

In Neuquén the larger and more regular vein deposits invite efficiency of operation, though problems inevitably increase with depth. Actually only two mines—La Riqueza, which is now closed, and San Eduardo which, with more equipment, could easily treble its present production—were competently developed. At Mina Escondida, which contains the chemically pure asphaltite or grahamite, the hoisting facilities can handle a scant 40 pct of working-face capacity if the mine were fully manned. At Mina Cura-Co haste in pulling

out the bitumen near the surface has left the new management with serious and expensive structural problems. In none of the properties is there evidence that mining is being carried on as a serious business, for technical direction and equipment are utterly inadequate. Apparently none of the owners has sufficient faith in the permanence of the industry to make appropriate investments in their properties.

Economics and Future of the Industry

There are historical reasons for this lack of faith: The asphaltite deposits have been known for many years, and 70 or more of them have been prospected. Their fuel value was quickly recognized, and some local use was made of the more accessible deposits; but the remoteness of the region, not only from Buenos Aires, but also from the railroads which penetrate western Argentina, rendered them economically unavailable until imports of British coal were cut off early in the war. Only the deposits of the Auca Mahuida district in Neuquén were consistently utilized, for the sole reason that their chemical purity and physical homogeneity gave them more valuable uses and a price substantially higher than the price of fuel.

With the acute fuel shortage which ensued when war broke out in 1939, rising prices brought the bitumens of Mendoza and Neuquén within range of the Buenos Aires market, and several new or revived mining operations were initiated between 1939 and 1942. With a price of 90 to 95 pesos f.o.b. Buenos Aires, and mining costs at 15 to 20 pesos, 70 to 80 pesos were available for transportation, overhead, and profit. To encourage the demand for domestic asphaltite and coal, the Government has required industrial consumers to purchase not less than 10 pct of their requirements from Argentinian producers of solid fuel and, at the same time, it gives them a 10 pct rebate on the price of all domestic fuel so purchased.

Obviously the industry is on such a precarious footing that some such guarantee of an outlet is vital for the few operators who have an established business. Their product was noncompetitive when British coal was available, and it will become noncompetitive again when foreign coal again enters the domestic market. Already South African coal is finding its way across the Atlantic. Although there has not yet been any important movement of Chilean coal into Argentina, access to some of Chile's production was one of the purposes of the recent trade agreement between the two countries. Competition from the recently reported subbituminous coal of southern Patagonia is not a serious threat; but, if imported coal enters the market in sufficient quantities to meet growing industrial demands, the wisdom of increasing or maintaining subsidies on Mendoza and Neuquèn asphaltite will become dubious.

Until the present time, the utility of the lower grade or chemically impure asphaltites for fuel and for the manufacture of gas comprises the only industrial potentialities which had received consideration, either by the producers or by the Government. There is reason to believe, however, that the material possesses chemical values, and that experimental research in this field may develop new possibilities and higher prices for the product. As already noted, the asphaltites with 60 pct or more fixed carbon yield 70 to 75 pct coke, 20 pct gas, with some tar, light oil, liquor, and ammonia. The gas has high Btu value, and the volume increases substantially in the asphaltites with lower fixed carbon content. Hydrogen and methane gas are its principal ingredients. The tar residue is high in the aromatics, naphthalene and anthracene, whereas the light oil is rich in benzene, with some toluene. Although this ensemble of products differs in kind and in quantities from the series associated with the coal tars,

it should, nonetheless, furnish a good base for a chemical industry. In the same region, moreover, there are several inorganic raw materials; sulphur is now being mined at the head of Río Atuel at Volcán Overo. Sodium chloride is being recovered from a small salina near the San Rafael-Malargüe road, and much more is available in the Salina Llancanelo—the closed depression into which Río Malargüe drains. Two gypsum horizons in the stratigraphic section provide limitless quantities of calcium sulphate. In addition to the energy available in the asphaltite and in local oil pools, there are excellent power sites on Río Diamante, Río Grande, and upper Río Neuquén.

There are many practical reasons for investigating the feasibility of a chemical industry in this part of Argentina. The Andean foothills and piedmont have a small population, and the scattered and modest mining operations constitute an important adjunct to the meager regional economy. Furthermore, they have been of some importance in bringing population and improved transportation and communication facilities into southwestern Mendoza and western Neuquén. The geologic character of the asphaltite deposits is such as to limit severely the size of any one operation; and if all the deposits with known or probable commercial tonnages were in production, it is unlikely that gross output would exceed 20,000 tons a month. As a supply of fuel, this amount is negligible; as a supply of raw material for a local chemical industry, on the other hand, it would be much more significant. With the prospect of asphaltite as fuel succumbing to competition from imported coal, a diversified chemical industry offers operators not only the prospect of staying in the mining business but also the possibility of a higher monetary return, not to mention a far more efficient industrial use for their product. It likewise offers Argentinians the possibility of establishing a new industry and of developing a thinly settled segment of their frontier.

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The field survey upon which the preceding report is based was one of the most comprehensive examinations which the solid bitumens of western Argentina have received. There was little need, therefore, to utilize older literature on the subject, even in regard to the chemistry of the deposits, for C. A. S. Piscione, who is responsible for much of the recent chemical work, was a member of the party. The following list of titles, to which reference is made in the text, merely directs attention to a few workers whose studies merit recognition, and to articles from which a longer list of references can be obtained.

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Occurrence of Heavy Minerals in the Pebble Phosphate Deposits of Florida

By Frank R. Hunter*

(New York Meeting, February 1948)

Introduction

Scope of Work

This paper represents the results of an investigation of the presence, amounts, and degree of concentration of heavy minerals found in the pulp of the phosphate flotation plant at Peace Valley, Fla., which is operated by the International Minerals and Chemical Corporation of Chicago.

In this paper, heavy minerals are those, excepting collophane, that sink in acetylene tetrabromide with a specific gravity of 2.95.

Percentages of heavy minerals at several points of the flowsheet and the species of minerals have been determined. Some information as to the screen size of the minerals has been acquired.

Previous Work

All previous published reports or investigations of the deposits of heavy minerals in Florida have been limited to present beach sands. The mining operations conducted by the Humphreys Gold Corp. about seven miles east of Jacksonville along an ancient shore line have been described in trade journals, but descriptions of the minerals have been lacking. Commercial grades of ilmenite, zircon, and rutile are produced on a large scale. Papers have been published on heavy minerals found on the Florida beaches.[1–6]

Location

"International's" Peace Valley flotation plant is in south-central Polk County, Fla. about eight miles south of Bartow and along the Atlantic Coast Line Railroad. The mine surrounds the plant and includes about 1500 acres of the phosphatic lands in this area.

Geology

General Setting

All of the rock underlying the area is almost horizontal. The oldest bed exposed is part of the Hawthorn phosphatic limestone of Miocene age. This outcrops in the stream bed of the Peace River about one mile east of the plant. Elsewhere the Hawthorn is covered except where it has been exposed by mining operations.

The Bone Valley formation of Pliocene age, which is the phosphatic stratum extensively mined in this area, unconformably overlies the Hawthorn.

Resting unconformably upon the Bone Valley formation are Pleistocene (?) terrace deposits of unconsolidated quartz sand which comprise the overburden of the phosphate ore.

Stratigraphy

Hawthorn formation: "Bedrock" of the pebble phosphate deposits is a cream to white colored, sandy, dense limestone containing varying amounts of small grains and nodules of collophane. The upper surface of the formation is undulating and irregular showing evidence of former

Manuscript received at the office of the Institute Nov. 10, 1947; revised May 20, 1948. Issued as TP 2456 in Mining Technology, September 1948.

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stream valleys and sink holes. The thickness of the Hawthorn here is not known definitely but probably exceeds 50 ft.

Bone Valley formation: The Bone Valley formation is principally an argillaceous sand, colored gray, cream, and brown and containing 20 to 40 pct of collophane (?). At its base in many places there occurs a coarse phosphatic conglomerate with pebbles up to 1 or 2 in. in diameter. Upward the rock has progressively finer textures and in places the sand becomes very argillaceous. Layers of a bluish green clay less than 1 in. thick are common in occurrence and are found in some rather extensive beds up to 3 ft in thickness. The total thickness of the formation is quite variable ranging from less than 15 ft to over 50 ft. At the Peace Valley mine it averages about 25 to 30 ft in thickness.

Pleistocene (?) sand: This sand is fine to medium grained and subangular in texture. It is composed of 90 to 95 pct quartz and 5 to 10 pct feldspar and is almost snow-white in color. It is sharply delineated from the underlying Bone Valley phosphatic sand and is thought to be in unconformable contact. The top of the Pleistocene (?) sand is the present land surface. Thickness of this surface sand varies from 0 to about 50 ft and is very irregular depending largely upon the extent of present erosion at any locality.

Heavy Mineral Deposits

Distribution

Thorough work on the actual field occurrence of heavy minerals in the Bone Valley formation has not been carried out except to determine average percentages. The percentages of heavy minerals, coarser than 325-mesh, in core samples taken at eight scattered localities in adjacent areas of the formation were as follows: 1.15, 0.67, 0.66, 0.47, 0.36, 0.28, 0.62 with an arithmetic average of 0.61 pct.

In the Bone Valley formation at the mine the distribution of the "heavies" entering the flotation plant were determined by samples caught every 8 hr over a 12-day period. The percentages of "heavies" vary from 0.10 to 0.56 and average 0.34. Before entering the flotation plant the rock has undergone the following operations which tend to mix the pulp: dug with dragline, pumped about one mile hydraulically, ⅜ in. (about 2.75 mesh) pebble removed by screens, −325-mesh material deslimed by hydroseparators, and +35-mesh material removed by Fahrenwald classifiers.

It is noteworthy that since the percentages are very small, variations of small amounts are important and chances for error are large. In the feed, concentrate, and tailing of the amine section the variation in percentages of "heavies" is somewhat greater, but also the percentages of heavies are several times greater, being on the order of 1 to 5 pct.

Ore mined is not selected in any manner for heavy mineral content. From the above it has been concluded that the heavy mineral content of the phosphate ore is fairly uniform over large areas and in large volumes, but with many small local concentrations.

Character

So-called heavy mineral deposits here are really nothing more than an accessory group of minerals in a phosphatic sand. The sands are unconsolidated and are made up by weight on the average approximately of: quartz, 46 pct; phosphate nodules, 30; clay minerals, 20; feldspar, 4.

The "matrix," as the ore is called, varies somewhat in size distribution of particles, but in the Peace Valley mine is rather uniform. The ratio of +⅜ in. (about 2.75-mesh) material to −⅜ in. is approximately 1 to 6 by weight. A typical sieve analysis by weight is:

Mesh	Weight, Per Cent
−2.75 + 35	24.1
−35 + 48	22.8
−48 + 65	42.5
−65 + 100	5.7
−100	4.9
	100.0

Mineralogy

Minerals Present—Mineralogical research has not been completed in detail, but the following minerals which sink in acetylene tetrabromide have been determined.

Ilmenite: Always a major constituent. Usually coal black but often a brownish color with a semimetallic luster on a finely pitted surface. It is notable that the specific gravity of this mineral is less than that of the zircon present.

Leucoxene: Very few complete grains. Often seen in small pits in the surface of ilmenite grains.

Zircon: Like ilmenite, is ubiquitous and usually the major component, especially in the finer sizes.

Rutile: Nearly always present in considerable amounts but varies more in quantity than ilmenite or zircon. Three gradational types are noted; a black almost opaque mineral, a dark fox-red type, and a lighter, red-orange colored mineral.

Staurolite: The most abundant of the uneconomic minerals of the heavies. Varies from yellow to brown in color. It seems to be rather consistent in amount.

Spessartite: At least three garnets have been seen, but this one is the most abundant by far. This mineral is extremely variable in abundance, sometimes making up a major portion of the heavies and again being almost entirely absent. It appears salmon pink under the microscope.

Almandite: This garnet is rather common but does not make a large percentage of the total. It is typically colorless.

Grossularite: Only occasional grains of this garnet have been noted.

Sillimanite: A very distinctive and persistent mineral sometimes becoming a major component of the heavy assemblage.

Kyanite: Always present and usually in moderate amounts.

Epidote: Always present but in quite small amounts.

Tourmaline: Quantity quite variable but always present, ranging from a trace to 5 or 6 pct of the total of the heavies.

Hornblende: Another variable mineral. Usually present in small amounts, but may make up 3 to 4 pct, or may be absent entirely.

Corundum: Usually present in very small amounts or traces.

Biotite: Quite variable in amount. Is usually almost entirely absent, but may make up an appreciable amount of the whole.

Sphene: A ubiquitous mineral but in small quantities.

Monazite: Almost always present but always in small amounts and sometimes only in traces.

Xenotime: Always present but in extremely small amounts.

In addition to the above, there are several minerals which occur only in small amounts or traces or have not been satisfactorily identified. These include: topaz, spinel, andalusite, wallastonite, and eudialite. Magnetite is apparently completely absent.

Variation of Heavy Minerals—Sufficient mineral counting to determine accurate averages of relative percentages of each of the heavy minerals has not been carried out. Actually this would be a tremendous task because of the great variation of even the more persistent minerals.

Grain counts of 2014 heavy mineral particles gave the following averages for the three economic minerals: ilmenite, 20 pct; zircon, 21; rutile, 7.5

Counts of a few hundred heavy mineral grains gave the following averages for the less abundant minerals: staurolite, 19 pct; sillimanite, 8; epidote, 2; tourmaline, 3; garnets, 2; hornblende, 1; corundum, 1; biotite, 2; kyanite, 1; monazite, 1; titanite, 2; and traces of topaz, andalusite and xenotime.

Grain Size—A measurement of several hundred grains of the heavy minerals showed that the average maximum diam-

eter is slightly greater than 150 microns or 100-mesh.

In the heavy mineral sampling of the plant it is very evident that most of the heavies are −65-mesh and also evident that the finer the grain size the greater the concentration of heavies. This is particularly true of zircon. In a −150-mesh sample of heavy minerals at least 50 pct is zircon. The total amount of the finer sizes of sand, however, is much smaller than that of the larger sizes. Table 1 shows the relationship between screen sizes and percentage of heavies in a sample taken from the amine feed of the Peace Valley plant.

TABLE 1—*Relationship between Screen Sizes and Percentages of Heavies*

Screen Size	Weight of Total Sample, Per Cent	Weight of Heavies of Screen Fraction, Per Cent	Weight of Heavies of Total Sample, Per Cent
−2.75 + 35	18.74	0.20	0.0375
−35 + 48	13.98	0.07	0.0098
−48 + 65	39.60	0.67	0.2650
−65 + 80	6.65	1.37	0.0911
−80 + 100	10.12	1.77	0.1790
−100 + 150	8.08	6.80	0.5500
−150	2.80	15.00	0.4200
	99.97		1.6188

CONCLUSIONS

1. The heavy mineral content of the Bone Valley formation in the vicinity of the Peace Valley plant is small and zones of marked concentration are numerous but neither large nor extensive.

2. From an economic standpoint the ratios of ilmenite, rutile, and zircon, to the rest of the heavy mineral assemblage compares favorably with many other known deposits.

3. Because of large tonnages processed and some concentration of heavies in the flotation-plant flowsheet it is possible that the heavy minerals might be removed economically.

ACKNOWLEDGMENTS

Research on this project was made possible by the full cooperation of the entire staff of the Research Division of International Minerals and Chemical Corporation and the writer wishes to express his gratitude especially to Dr. I. Milton LeBaron, Supervisor of Phosphate Research, Mr. Franklin Farley, Vice President, Dr. Paul D. V. Manning, Vice President in Charge of Research, and Mr. Louis Ware, President.

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DISCUSSION

O. C. RALSTON*—These heavy minerals accessory to the phosphatic sand mined in the pebble district of Florida are of interest because the gross tonnage moved daily is great. The heavy minerals are not present in sufficient amount to be mined for themselves alone but they are present in sufficient quantity to make their recovery worthwhile as an additional product normally wasted. With this in mind it is well worthwhile to determine the amounts and grades of the various concentrates that can be recovered. If 20 pct each of ilmenite and zircon can be expected from the heavy family and 7.5 pct rutile, it is apparent that nearly half of the heavies is probably marketable The 1 pct monazite must not be forgotten. Of these I look for the ilmenite to find the most stable market, zircon to become more highly competitive within a few years, and rutile is already a competitive mineral and no longer strategic. The growing importance of the titania pigment industry and the promise of metallic titanium as a future industry makes the titanium minerals in this material of first importance. This discovery has added an important item to Florida's already important titanium and zirconium resources.

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A Study of Opaque Minerals in Trail Ridge, Florida Dune Sands

By E. E. Creitz* and T. N. McVay,* Member AIME

(New York Meeting, February 1948)

Introduction

Object

Rather large amounts of titanium minerals and some zircon and monazite are being recovered from dune sands about 10 miles west of Jacksonville Beach, Fla. The Mining Branch of the Bureau of Mines, Southern Experiment Station, has been interested in developing additional large deposits of similar sands in other areas of Florida. The results of a field study have been published[1] concerning the sands occurring on Trail Ridge which runs north and south about 30 miles southwest of Jacksonville.

This investigation was undertaken in order to identify the opaque titanium minerals occurring in Trail Ridge and because of the economic importance of Florida titanium minerals.

Nature of the Problem

The minerals present in the heavy fractions (sp gr > 2.94) of the sands are principally opaque minerals; staurolite, zircon, sillimanite, tourmaline, kyanite, rutile, a small amount of corundum, traces of the zinc spinel gahnite and garnet. Epidote, monazite, hornblende and titanite are present in other dune sands but these minerals were not noted in the samples collected on Trail Ridge.

All of the minerals except those which are opaque can be readily identified by the usual petrographic methods. However, opaque minerals cannot be examined satisfactorily by transmitted light. Polished sections of the opaque-mineral grains were examined but these sections did not furnish useful information. Consequently, it was necessary to use means other than the usual petrographic methods to identify them.

Previous Investigations of Heavy Minerals in Florida Sands

Martens[2] has described the heavy minerals in the Florida beach sands and has classified the opaque minerals as ilmenite with minor amounts of leucoxene.

Miller[3] studied a number of sands collected by Ross and Mertie in Florida. He states that the material from the Florida beach and dune sands that commonly has been called ilmenite has essentially the chemical composition of the material that has been called arizonite.[4]

The X ray diffraction patterns of several samples indicated that the supposed ilmenite is essentially amorphous, with only weak lines corresponding to ilmenite, and hence the material is unlike the type of arizonite from Arizona.

Opaque Titanium Minerals

Rutile is essentially titanium dioxide. However, as pointed out by Dana,[5] important amounts of Fe^{+2} and Fe^{+3} as well as Cb and Ta are reported in some analyses,

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Published by permission of the Director, Bureau of Mines, U. S. Department of the Interior, Washington, D. C. Manuscript received at the office of the Institute Jan. 30, 1948. Issued as TP 2426 in Mining Technology, July 1948.

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[1] References are at the end of the paper.

Smaller amounts of Sn, Cr and V may be present. Reported analyses of rutile always show the presence of impurities. In transmitted light, rutile is usually red or brown-red and the depth of color increases with increasing amounts of Fe^{+3}. The columbian and tantalian varieties are deep brown or opaque. The ferrian variety known as nigrin is opaque in thick sections or grains.

The petrographic examination is best accomplished by using transmitted light with crossed nicols and with the condensing lens above the polarizer swung into position to give convergent light. Rutile appears clear and transparent. Grains that appear to be opaque by parallel light frequently become transparent when the technique described above is used. The specific gravity of practically pure TiO_2 as given by Dana[5] is 4.23 and other authorities are essentially in agreement. However, Dana[5] gives data to show that there is some variation in the specific gravity of rutile depending upon the composition.

Ilmenite[5] is essentially iron, magnesium, manganese and titanium dioxide with as much as 54 atomic per cent of magnesium, and grading into geikelite, (Mg·Fe) TiO_3, with Mg:Fe as 8:1. Pyrophanite is $MnSiO_3$ with Mn:Fe as 5:7 in some analyses. Only limited amounts of Fe_2O_3 can enter into the composition of ilmenite at ordinary temperatures and more than 6 pct by weight is present, presumably as admixed hematite or magnetite. Excess TiO_2 reported in some analyses may be caused by admixed rutile, which is known to occur with ilmenite in intimate mixture. Ilmenite is opaque or nearly so. According to Dana[5] ilmenite has a specific gravity of 4.72 ± 0.04; Winchell,[6] 4.8 to 4.9; Milner,[7] 4.5 to 5.0; Krumbein and Pettijohn,[8] 4.6 to 4.9.

Leucoxene is defined and described by Dana[5] as follows:

> A name loosely applied to dull, fine-grained yellowish to brown alteration products high in titania. Found as an alteration product of sphene, ilmenite, perovskite, titanian magnetite or other titanium minerals. The material consists in most cases of rutile, also, less commonly of anatase or sphene.

According to Krumbein and Pettijohn[8] the composition and crystallization of leucoxene is uncertain. An unaltered core of ilmenite may be present. They give the specific gravity as 3.5 to 4.5. Milner[7] describes the mineral in about the same manner as Dana, although he mentions that leucoxene is nonmagnetic except when the grains have an unaltered core of ilmenite and then it is weakly magnetic. According to Ross[9] the leucoxene in the Roseland area of Virginia is sphene, and the leucoxene associated with the Magnet Cove rutile is anatase.

Tyler and Marsden[10] examined a number of leucoxene samples and reported that in most instances leucoxene consisted of rutile.

Arizonite was named and described by Palmer[4] and has the formula $Fe_2O_3 \cdot 3TiO_2$. The larger particles of very fine powder are opaque but the very thin edges of minute slivers are deep red by transmitted light. The mineral has a high index of refraction with moderate birefringence. The arizonite alters to a meshwork of fine anatase and is then dull in luster and brownish yellow in color. The alteration product would be classed as leucoxene. The specific gravity is 4.25.

Specific gravities of the four minerals are approximately as follows: rutile, 4.23; ilmenite, 4.5 to 5.0; leucoxene, 3.5 to 4.5 and arizonite, 4.25. The data indicate that it should be possible to separate ilmenite from the other three minerals.

Methods of Heavy Mineral Separation

The first step in separating the heavy-mineral grains from quartz and other low specific-gravity minerals was by float and sink in acetylene tetrabromide (sp gr, 2.94). The sink fraction was further divided into magnetic and nonmagnetic fractions

by means of a strong electromagnet. Most of the opaque minerals were concentrated in the magnetic with a lesser amount in the nonmagnetic fraction. The magnetic portion constituted 73 pct by weight of the heavy mineral sample and contained opaque minerals, staurolite and tourmaline as the major constituents with traces of garnet and gahnite. A few grains normally nonmagnetic were present in this fraction. The nonmagnetic portion comprises 27 pct by weight of the samples and contained opaque minerals, zircon, sillimanite, kyanite, rutile, and a trace of corundum. Both the magnetic and nonmagnetic fractions were further separated by means of Clerici's solution. The method was substantially as outlined by Krumbein and Pettijohn.[8] An electrically heated constant-temperature bath was used for heating the Clerici's solution and the specific gravities were determined by means of pycnometers.

Leaching of Opaque Titanium Minerals and True Ilmenite

A method for leaching titanium from ilmenite by using hot concentrated sulphuric acid is reported to have been employed in Germany.[a] The procedure as outlined was for the treatment of ton batches of ilmenite. This method was modified by the senior author so that it could be used for leaching titanium ores on a laboratory scale. One gram of minus 400-mesh material was weighed into a 13 × 120 mm Pyrex test tube, 0.8 ml of concentrated sulphuric acid was added and the sample heated in a Wood's metal bath to 170–180°C for 10 min. The Wood's metal was wiped off while hot and the test tube with sample allowed to cool. The digested ore and sulphuric acid formed a cake at the bottom of the test tube. The test tube was cut off above the cake and the cake with the test tube was transfered to a large agate mortar. After 10 ml of 62 pct sulphuric acid was poured in the mortar, the cake was broken up carefully and mixed with the acid. The pulverized cake was transferred to a 250 ml beaker with an additional 15 ml of 62 pct H_2SO_4, heated to boiling and then digested overnight on a steam bath. The contents were transferred to a 250 ml volumetric flask which was filled to the reference mark with distilled water. Aliquot parts of this sample were analyzed colorimetrically for titanium.

[a] Communication from Alton Gabriel, College Park Station, Bureau of Mines.

Identification of Magnetic Opaque Heavy Minerals

The magnetic fraction comprising 73 pct of a large composite sample was separated into fractions by means of hot Clerici's solution according to the method outlined above. The data for the separations are given in Table 1.

Table 1—*Separation of Magnetic Fraction by Means of Clerici's Solution*

Fraction Number	Specific Gravity		Percent by Weight	Description of Sample
	Sink	Float		
1	2.94	3.31	16.2	Tourmaline[a]
2	3.31	3.90	41.4	Staurolite and opaque[b]
3	3.90	4.00	1.0	Opaque[c]
4	4.00	4.10	15.3	Opaque
5	4.10	4.35	24.7	Opaque
6	4.35		1.4	Opaque and zircon[d]

[a] Small amount of sillimanite.
[b] Contains about 40 pct opaque with a trace of kyanite.
[c] About 15 pct staurolite.
[d] Contains about ⅛ zircon and the balance opaque.

Tourmaline, kyanite, zircon and most of the staurolite fell within the proper specific-gravity fractions. Ilmenite with a specific gravity of 4.5 to 5.0 should fall in fraction 6. However, it is quite evident that very little if any true ilmenite is present. The opaque minerals are estimated to comprise about 58 pct of the magnetic fraction.

Chemical Analyses of Some of the Specific-gravity Fractions

The chemical analyses of some of the specific-gravity fractions of the magnetic heavy minerals are shown in Table 2.

TABLE 2—*Chemical Analyses of Specific-gravity Fractions of Magnetic Heavy Minerals*
PER CENT

	Fraction Numbers		
	2	3 and 4	5
SiO_2	20.80		
Fe_2O_3	14.64	19.26	24.87
TiO_2	24.50	74.20	68.80
Al_2O_3	34.46	0.94	1.13
MnO	0.04	0.50	0.26

Fraction No. 2 contained a large amount of silica and alumina and these oxides with part of the iron came from the staurolite. The titania was evidently present in the opaque mineral. The Fe_2O_3:TiO_2 ratio in the combined 3 and 4 fractions was 0.26 whereas in fraction No. 5 it was 0.36 Consequently, the heavier fraction had a higher Fe_2O_3:TiO_2 ratio. However, this ratio for ilmenite (Fe TiO_3) is 1.0 providing that all of the iron present is calculated as ferric oxide. The Fe_2O_3:TiO_2 ratio for arizonite is 0.67. Consequently, on the basis of chemical analyses the magnetic opaque heavy mineral is neither ilmenite nor arizonite.

Opaque grains from fraction 5 were hand picked according to color for spectrochemical analysis. The percentages of grains of different colors and the Fe:Ti ratio as determined by spectrochemical analysis are shown in Table 3.

TABLE 3—*Color of Grains and Fe:Ti Ratios in Fraction* 5

Color	Number of Grains in Sample	Per Cent	Fe: Ti Ratio
Yellow	34	6.7	0.25
Brown	338	66.7	0.33
Black	135	26.6	0.54

Although the determination of the Fe:Ti ratio is not precise the ratio was greater with increasing intensity of color.

Leaching Tests of Magnetic Opaque Minerals and Ilmenite

A sample of the magnetic, heavy-mineral fractions with a specific gravity between 3.88 and 4.31 was leached according to the method outlined above. Also ilmenite from Kragerö, Norway, and the so-called ilmenites from Travancore, India, and from Vero Beach, Fla., were given the same treatment. The results are shown in Table 4.

TABLE 4—*Results of Leaching Tests on Titanium Mineral Samples*
PER CENT

Description of Sample	TiO_2 in Sample	TiO_2 Leached	Total TiO_2 Removed by Leaching
Magnetic opaque[a]	67.5	24.4	36.0
Vero, Florida "ilmenite"[b]	59.2	35.4	60.0
India ilmenite[c]	52.3	45.6	87.0
Norway ilmenite	40.4	40.4	100.0

[a] The sample contained over 95 pct opaque minerals.
[b] Sample labeled ilmenite but X ray diffraction pattern indicates that it is leucoxene.
[c] Miller[3] reported that the Travancore, India, "ilmenite" is arizonite.

The results show that the magnetic opaque mineral in the Florida Trail Ridge sand does not leach as completely as ilmenite or the other samples tested.

X ray diffraction patterns were made on Norwegian ilmenite, the magnetic opaque mineral from Trail Ridge, Fla. and Polk County, Tennessee rutile. The samples were ground to the same fineness and diffraction patterns made by the use of a molybdenum target tube and a zirconium metal filter. The time of exposure was 4 hr. (The diffraction pattern of the nonmagnetic opaque and known samples of ilmenite and rutile are shown in Fig 1. The spots on the diffraction patterns of the rutile and ilmenite indicate a coarse crystalline structure, whereas, the diffuse lines of the magnetic opaque indicate a fine

crystalline structure.) The diffraction pattern of the latter, except for one missing faint line matches closely that of the rutile from Polk County, Tenn. The diffraction pattern of ilmenite differs significantly from that of the opaque mineral and rutile. Consequently, the opaque mineral is not ilmenite.

Conclusions: The opaque magnetic mineral present in the Trail Ridge, Fla. sand has a lower specific gravity, does not leach as well, and has a lower iron content than ilmenite. Also, it has the crystal structure of rutile, does not conform to the formula for arizonite, and has a considerable latitude in $Fe:TiO_2$ ratio; consequently the opaque mineral is classed as a leucoxene containing extremely fine-grained rutile.

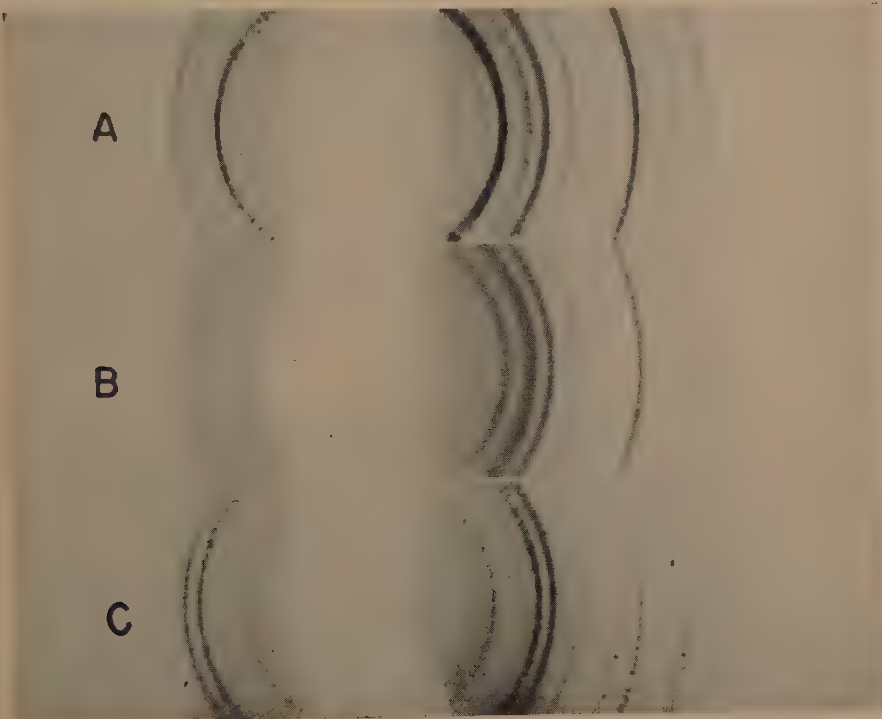

FIG 1—DIFFRACTION PATTERNS OF NONMAGNETIC OPAQUE AND KNOWN SAMPLES OF ILMENITE AND RUTILE.

A. Rutile, Polk County, Tenn.
B. Magnetic Opaque, sp gr 4.31 to 4.42, Trail Ridge Area, Fla.
C. Ilmenite, Kragerö, Norway.

IDENTIFICATION OF NONMAGNETIC OPAQUE HEAVY MINERALS

The nonmagnetic fraction constituting 27 pct of the total heavy mineral content was also separated in Clerici's solution as described above. The results are shown in Table 5.

TABLE 5—*Separation of the Heavy Minerals by Clerici's Solution*

Fraction Number	Specific Gravity		Per Cent by Weight
	Sink	Float	
1	2.94	3.90	36.7
2	3.90	4.10	9.3
3	4.10	4.42	4.3
4	4.42	4.60	2.4
5	4.60		47.3

The mineral composition determined by grain count is shown in Table 6.

TABLE 6—*Mineral Composition of Nonmagnetic Fractions by Grain Count*
PER CENT

Mineral	Fraction Number from Table 5				
	1	2	3	4	5
Sillimanite	68.4		0.9		
Kyanite	26.4	2.4	2.0	0.4	
Opaque	4.5	80.9	42.5	24.6	2.5
Rutile	0.7	8.1	51.5	24.2	1.5
Corundum		8.1	0.9	0.1	
Zircon		0.5	2.2	50.7	96.0

It is believed that the sink-and-float determinations in Clerici's solution are approximately correct, because the majority of the sillimanite, kyanite, corundum and zircon grains fall in the proper specific-gravity fractions. The rutile has a considerable spread in specific gravity but the bulk of the opaque material is in No. 2 fraction and has a lower specific gravity than most of the rutile. On account of the high concentration of the opaque mineral in fraction 2 (80.9 pct) this fraction was analyzed for its titania and iron content. A similar analysis was made on a sample of rutile from Polk County, Tenn. Leaching tests were made on these minerals and the results are also shown in Table 7.

TABLE 7—*Results of Leaching Tests and Chemical Analyses of Nonmagnetic Opaque Heavy Mineral and Polk County Rutile*
PER CENT

Sample	TiO_2	Fe_2O_3	TiO_2 Leached	Total TiO_2 Leached
Nonmagnetic opaque	86.9	7.6	6.6	7.5
Rutile	98.2	2.5	6.5	6.0

In so far as leaching is concerned, the two minerals are quite similar.

Samples of both the nonmagnetic opaque mineral and the Polk County, Tenn., rutile were ground to the same fineness and X ray diffraction patterns obtained. The patterns are shown in Fig 2. The patterns of the nonmagnetic opaque mineral and the Polk County, Tennessee rutile are similar but the diffraction lines of the nonmagnetic opaque mineral are more diffuse indicating rutile with an extremely small crystal size.

Conclusions: The results of the X ray diffraction patterns show that the nonmagnetic opaque mineral has a rutile structure, however, the crystal size is smaller than the usual rutile. Also, most of the opaque mineral grains have a lower specific gravity than rutile. The nonmagnetic opaque mineral may be classed as a leucoxene containing a large amount of finely crystallized rutile.

SUMMARY

Based on leaching tests, specific gravity determinations, and X ray diffraction patterns, the opaque minerals in both the heavy magnetic and nonmagnetic fractions in the Trail Ridge sands of Florida may be classed as leucoxenes having varying specific gravities and titania contents. Those which are magnetic have a high iron content, however, the amount of iron is less than that present in either ilmenite or arizonite.

The leaching tests show that the titania in the opaque minerals is much less soluble than the titania in ilmenite.

It is possible that the leucoxenes were formed by weathering of the ilmenite originally present in the minerals deposited in the Trail Ridge area of Florida.

ACKNOWLEDGMENTS

This report is one of many on the various aspects of the Bureau of Mines program directed toward the more effective utilization of our mineral resources.

Investigations of the mineral resources are conducted by the Mining Division, L. B. Moon, Chief, and by the Metallurgical Division, O. C. Ralston, Chief.

The scope of this project falls in the

province of the Metallurgical Division, whose activities embrace the separation of difficultly beneficiated ores, the production of pure metals from domestic deposits, the exploitation of marginal ore reserves, and the improvement of present industrial metallurgical practice.

The authors wish to acknowledge the assistance of the Ore Dressing Section and the Mining Branch of the Southern Experiment Station for their cooperation. Thanks are due to H. G. Iverson, Chief, Tuscaloosa Branch, Metallurgical Division, Southern Experiment Station for his encouragement in promoting this study and for his assistance in the preparation of the manuscript.

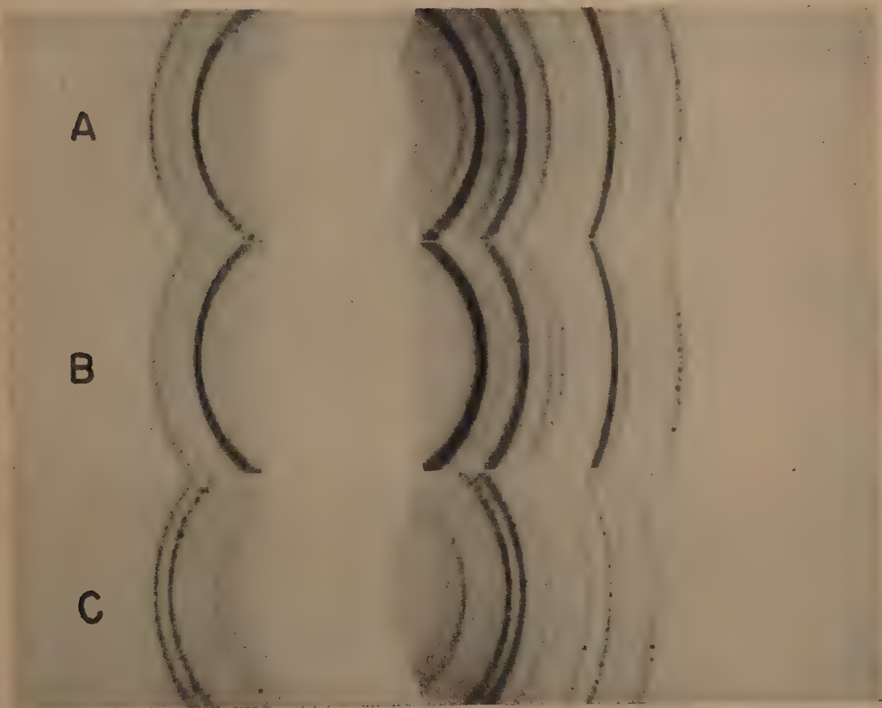

FIG 2—DIFFRACTION PATTERNS OF NONMAGNETIC OPAQUE MINERAL AND POLK COUNTY RUTILE
A. Rutile, Polk County, Tenn.
B. Nonmagnetic opaque, sp gr 3.90 to 4.10, Trail Ridge Area, Fla.
C. Ilmenite, Kragerö, Norway.

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Recent Developments in Mining, Processing, and Application of Nepheline Syenite from Blue Mountain, Ontario

By H. R. Deeth* and C. J. Koenig*

(New York Meeting, February 1948)

About ten years ago nepheline syenite was introduced to the ceramic industry and the material has now found application in the various branches of the industry, namely, as a vitrifying agent in whitewares and as a source of alumina and alkalies in glasses, glazes, and porcelain enamels. Spence[1] described the early mining and processing developments regarding this material. Since the time of his article many research papers have appeared relating to the usage of nepheline syenite but little has been published relative to subsequent mining and processing methods. A complete indexed review of the literature was issued recently by Ohio State University.[2]

Since "Lakefield" nepheline syenite from Blue Mountain, Ontario (Fig 1), is the only commercial ceramic-grade nepheline syenite now being marketed on this continent, discussion is limited to this operation carried on by American Nepheline Limited, Lakefield, Ont., and its subsidiary in the United States, American Nepheline Corp. at Rochester, N. Y.

The Deposit

Blue Mountain is a long ridge rising to an elevation of about 350 ft at its highest point above the surrounding country and extending for about six miles in a northeasterly direction across Methuen township, Peterborough County. The rock of the Blue Mountain central alkaline intrusive is fine grained, light colored and of even granitic texture. The solidified mass consists predominately of the minerals albite, nepheline and microcline in order of abundance. The average mineralogical composition for the deposit obtained by a study of 112 samples taken regularly over the whole formation is as follows:[3]

Mineral	Volumetric Per Cent
Albite	54
Microcline	20
Nepheline	22
Muscovite	2
Mafics (Biotite, hastingsite, magnetite)	2
Total	100

The mineral nepheline (or nephelite) belongs to the feldspathoid group but is lower in silica content than are the feldspars. It is a silicate of soda, potash, and alumina ($NaKAl_2O_32SiO_2$). Its presence with albite ($Na_2O\ Al_2O_3\ 6SiO_2$) and microcline ($K_2O\ Al_2O_3\ 6SiO_2$) gives the Blue Mountain deposit an alkali content of 15 pct and an alumina content of approximately 24 pct.

While this huge alkaline mass is somewhat variable mineralogically, it has a uniform chemical composition. When the iron, mostly in the form of magnetite, is removed during processing, a remarkably uniform finished product is obtained.

Geological surveying has disclosed many million tons of nepheline syenite in the Blue Mountain deposit. Diamond drilling in the area where mining is now being carried out indicates a block of about one million tons.

Manuscript received at the office of the Institute Feb. 6, 1948. Issued as TP 2406 in Mining Technology, July 1948.

* American Nepheline Corp., Rochester, N.Y.

[1] References are at the end of the paper.

The average chemical analysis of the finished product is as follows: silica (SiO_2), 60.24 pct; alumina (Al_2O_3), 24.00; iron (Fe_2O_3), 0.07; calcia (CaO), 0.15; magnesia (MgO), 0.02; soda (Na_2O), 10.03; potash (K_2O), 5.01; loss on ignition, 0.46; total, 99.98. The analysis of the crude rock is similar except that the iron content is approximately 1.50 pct.

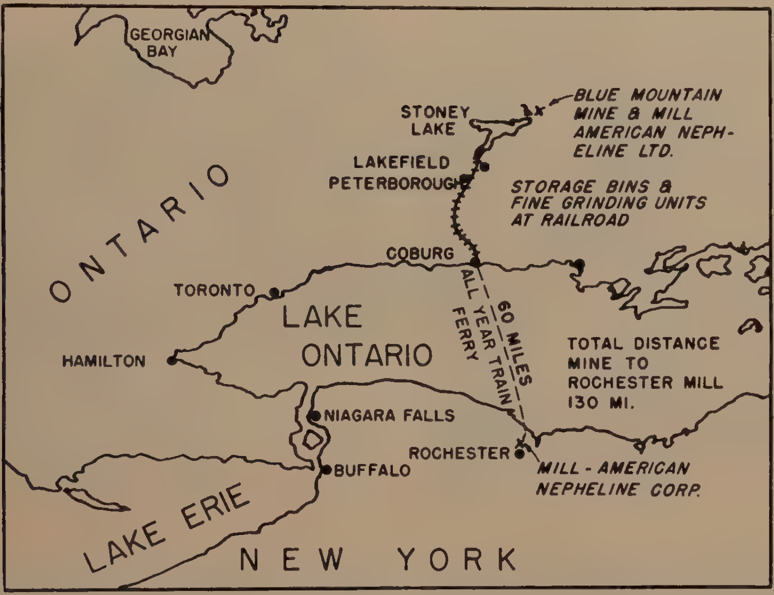

FIG 1—LOCATION OF NEPHELINE SYENITE DEPOSITS AND MILLS.

MINING—UNDERGROUND AND OPENCUT

Rock is mined underground by the shrinkage-stoping method. The stoping area now being worked consists of two stopes, each about 50 ft wide, one being 170 and the other 150 ft long. Ore cars are loaded through box holes and chutes at the bottoms of the stopes. This is illustrated in Fig 2. The ore cars are hauled by a battery locomotive to a grizzly spaced at 14 in., through which rock is dumped into a bin for loading the skips (Fig 3). The loaded skips are hoisted through a 300-ft inclined shaft driven into the side of Blue Mountain at a dip of 10°. The contents of the skips are dumped into the jaw-crusher feed bin.

During the summer months opencut mining (Fig 4) is carried out in the same vicinity as the underground workings. The face is about 60 ft high and is blasted down after it has been drilled in 20-ft benches. The rock blasted down is subjected to secondary blasting when necessary and loaded into trucks by a Diesel shovel having a one-yard bucket. This rock is trucked about 1000 ft to the mouth of a raise (Fig 5), which comes through from the underground workings to the top of Blue Mountain, a distance of 170 ft. The rock dumped into this raise can be fed to the same grizzly that receives the underground ore. The use of two different methods of mining results in a very flexible operation. This system was put into operation during 1947 to supply rock for the new mill constructed at the mine site.

NEW MILL AT BLUE MOUNTAIN

The coarse-crushing plant is at the mouth of the inclined shaft on the side of Blue Mountain (Fig 6). It employs a 24 by 36-in. jaw crusher that takes 14-in. feed and re-

duces it to about 4 in. in size (Fig 7). The crushed rock is then transported through a conveyor gallery to the mill. At the discharge end of the conveyor a suspension magnet is placed over the belt to remove all forms of magnetic material, such as drill bits, nails, and other mine refuse. Two men stand on either side of the conveyor, which has a 36-in. belt, to watch the rock and pick off foreign material that may not be removed by the magnet.

The 4-in. rock from the jaw crusher is fed directly from this belt to a 3-ft standard cone crusher, where it is reduced to approximately ¾-in. size. This rock is fed to a 6 by 45-ft, direct-type, oil-fired rotary drier, as it is necessary that all moisture be completely removed from the rock to facilitate magnetic separation and fine screening.

From the drier the minus ¾-in. rock is elevated to a screening station where

2

3

Fig 2—Loading mine cars through chutes from stopes.
Fig 3—Dumping ore cars into loading bin for skip cars.

4

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Fig 4—Opencut mine on top of Blue Mountain.
Fig 5—Dumping ore from opencut mine into 170-foot raise.
This ore joins the ore from the underground mine at the loading bin and is hoisted up the inclined shaft in skip cars to the crushing plant.

24-mesh material is taken out and conveyed to screens over the magnetic-separator storage bins. The coarse fraction is dropped into a storage bin and fed to a 3-ft Symons short-head cone crusher. The discharge from this crusher is conveyed and elevated to a screen station over the magnetic-separator bins and classified on a 28-mesh screen. The minus 28-mesh material drops through into the storage bins after passing over a magnetic drum and is used as feed for the magnetic separators. The plus 28-mesh can be returned by conveyor to the 3-ft short-head cone crusher or sent to a set of 4- by 24-in. crushing rolls for further grinding and returned to the same screening station over the magnetic-separator storage bins.

The high-intensity magnetic separators are fed 28-mesh material from the storage bins. The magnetic portion is removed and discarded, at the present time. Metallurgical test work is being done to develop a process for recovering part of this waste from the magnetic separators. The finished product is conveyed to storage bins after passing through a scalping screen to ensure against oversize particles remaining in the final glass-grade product. The magnetic separators reduce the iron from about 1.50 to approximately 0.07 pct.

Dust control in the mill is maintained by cyclone dust collectors.

Electric power for operations at Blue Mountain is obtained from the Hydro Electric Power Commission. The high-tension line from the plant at Healy Falls, Ont., a distance of 30 miles, was completed in 1947.

This mill at Blue Mountain has been designed to leave adequate room for future expansion.

Transportation and Storage

The finished glass-grade material is trucked in covered tank trucks from the mill at Blue Mountain to the railroad at Lakefield, a distance of 26 miles, to concrete storage bins (Fig 8) containing approximately 4000 tons.

The mill at Rochester, N. Y., requires a ¾-in. crude rock. This rock is hauled from

Fig 6—New mill at Blue Mountain.

the mine by truck 4 miles to the northeast end of Stoney Lake and dumped on barges (Fig 9), which carry an average load of about 350 tons. These barges are taken by tug a distance of 22 miles to the dock at Lakefield, which is on the railroad. Here the barges are unloaded by a gasoline crane (Fig 10) with a clamshell bucket and placed in open hopper cars for shipment to the Rochester plant or put on a stock pile adjacent to the railroad. Sufficient rock is barged down during the navigation season from May to November and stock-piled to maintain shipments to the Rochester plant during the months when navigation is suspended. If necessary, ¾-in. rock can be hauled direct from the mine to the railroad cars during this period. The cars of crude rock are shipped to the Rochester plant via the Ontario car ferry, which operates all year round from Cobourg, Ont., to Rochester, N. Y.

Fine-grinding Operations at Lakefield

The production of 200-mesh pottery grade is carried out in a building connected with the storage bins at Lakefield. The 28-mesh glass-grade product is drawn from storage and fed to the fine-grinding unit, which is an 8 by 4-ft Hardinge pebble mill in conjunction with an air classifier to

which the feed is controlled by an electric ear. This system produces a uniform grind with specifications of not more than 2.5 pct on 200-mesh and not less than 95 pct through 325-mesh. The finished product is then placed in the storage bins.

Mill at Rochester, N. Y.

At the mill at Rochester, N. Y. (Fig 11), the ¾-in. crude rock received from Lakefield in hopper cars is unloaded into a track hopper, which feeds it to a conveyor belt that carries it into storage bins having a

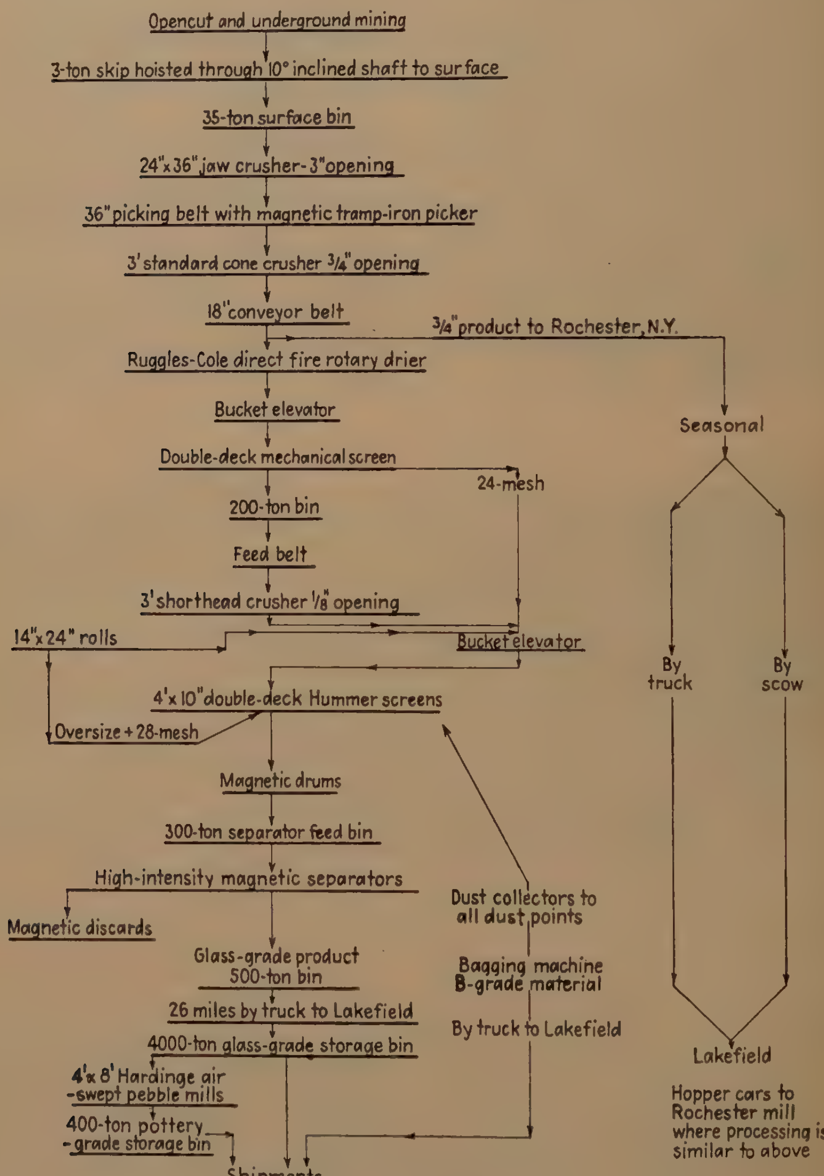

Fig 7—Flowsheet of new mill.

capacity of 2500 tons. The rock is drawn from the bin as required and if necessary is passed through a 5 by 40-ft direct-type, oil-fired rotary drier. From the drier the the pebble mill is passed over a magnetic drum and thence to a 28-mesh screen. The minus 28-mesh material goes to storage bins for magnetic separation. The oversize

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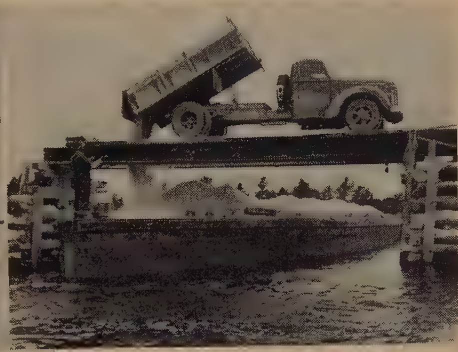

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FIG 8—CONCRETE STORAGE BINS AT LAKEFIELD, ONTARIO—CAPACITY 4000 TONS.
FIG 9—LOADING BARGE AT STONEY LAKE.

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11

FIG 10—TRANSFERRING ¾-INCH ROCK FROM BARGE TO RAILROAD CARS OR STOCK PILE FOR ROCHESTER MILL.
FIG 11—AERIAL VIEW OF MILL AT ROCHESTER, N. Y.

¾-in. rock is elevated to a double-deck mechanical screen equipped with ⅛-in. and 24-mesh screens. The plus ⅛-in. material is fed through a bin to a 3-ft short-head cone crusher and the discharge is in closed circuit with the screens. The minus ⅛-in. plus 24-mesh rock is fed through a bin to a 36-in. by 8-ft Hardinge pebble mill, close-circuited with its own screening station using a 24-mesh screen. The minus 24-mesh material from the drier discharge, the Symons short-head cone crusher and from this screen is in closed circuit with the pebble mill for further grinding. The high intensity magnetic separation (Fig 12) is the same at this mill as it is at the mill at Blue Mountain. The magnetic waste is discarded, and the finished product is conveyed to storage bins.

There are two fine-grinding units at this plant similar to the one at Lakefield and they are fed automatically by an electric ear (Fig 13). A pottery-grade material is made having the same specifications as to

size and iron content as the product at the Lakefield plant.

Dust control at the Rochester plant is maintained with a large Sly bag-type collecting system.

FIG 12—HIGH-INTENSITY MAGNETIC SEPARATORS AT RIGHT AND DUST COLLECTORS AT LEFT.

MINE AND MILL CAPACITIES

Present average capacity of the mine is approximately 9000 tons per month. This is increased during months of navigation by supplementing material from opencut mining.

The capacity of the Blue Mountain mill is approximately the same as that of the Rochester mill, which is about 3000 tons of glass-grade product per month. At Rochester 1300 tons of this glass grade is further ground for pottery grade. At Lakefield 650 tons is ground for pottery grade. Another fine-grinding unit of the same capacity is being installed at Lakefield. As a net result, tonnage for the ceramic industry per month is as follows: 4050 tons available for glass plants and 1950 tons for pottery plants. An additional 650 tons of pottery grade will be available when the new unit at Lakefield is in operation.

Besides this tonnage, there is also available from both mills approximately 600 tons per month of a grade B product, which contains about 0.6 pct iron. This product is slightly coarser than the 200-mesh pottery-grade material.

INDUSTRIAL APPLICATIONS

In the whiteware industry, nepheline syenite is being used both as a body and as a glaze ingredient. In bodies it is used to contribute to the glassy phase that binds

FIG 13—FINE-GRINDING UNIT CONTROLLED BY ELECTRIC EAR FOR PRODUCTION OF 200-MESH POTTERY-GRADE MATERIAL.

the other constituents together and gives strength to the ware. In glazes it serves as a readily fusible material for introducing the alkalies, alumina and silica into the formulation. The whiteware industry includes sanitary porcelain, high-tension and low-tension electrical porcelain, semivitreous dinnerware, hotel china, fine translucent china, dental porcelain, floor and wall tile, art pottery, porcelain liners and balls, and kitchenware.

The lower fusibility and greater fluxing action of nepheline syenite as compared with that of the traditional vitrifying agents enables a manufacturer to either fire the ware at a lower temperature or use a reduced amount of nepheline syenite and still attain the desired properties. Using a reduced amount of the vitrifying agent, which is nonplastic, allows for the presence

of more clays that provide the plasticity needed to form ware.

Nepheline syenite is being used in the various types of glasses that contain alumina and alkalies in the batch. The low iron content (0.06 to 0.07 pct Fe_2O_3) combined with the high alumina content makes it a desirable means of introducing alumina, especially where low iron is important. Opal glasses containing appreciable percentages of nepheline syenite are being produced. The low fusion characteristics and the relatively low thermal expansion of nepheline syenite in the fused state are of particular significance in glasses embodying high percentages of the material.

In enamels, nepheline syenite is used as a frit ingredient in ground and cover coat for sheet steel and cast iron. B-grade material (0.6 pct Fe_2O_3), which sells for about the price of flint, is also being used in ground-coat frits.

Nepheline syenite is used as a mill addition material for ground-coat enamels. The fact that the thermal expansion of nepheline syenite, in both the crystalline and glassy states, is near that of typical ground-coat frits is an important factor in this application.

Miscellaneous industrial adaptations of nepheline syenite include its role as a fired bond in abrasive grinding wheels and as an intermediate bond for refractory cements. The B-grade material is used as a vitrifying agent in some types of structural or heavy clay products.

Acknowledgment

The writers wish to express their appreciation to G. A. Bole, of The Engineering Experiment Station at Ohio State University, and to C. L. Cruikshank, of the Great Lakes Foundry Sand Co., for their helpful advice.

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Past Progress of Mineral Industry Education

By L. E. Young,* Member AIME

(New York Meeting, March 1947)

The progress of mineral industry education will be limited to the period prior to World War II and will be considered as primarily a division of engineering education. Its relation to progress in the mineral industry and to the activities of the American Institute of Mining and Metallurgical Engineers will be emphasized.

The history of mineral industry education has been recorded most completely and authoritatively by Dr. T. T. Read in his recent volume in the Seeley W. Mudd Series of American Institute of Mining and Metallurgical Engineers publications, entitled, *The Development of Mineral Industry Education in the United States*. Therefore, most of the historical details will be omitted from the discussion and an effort will be made to point out significant developments and to analyze trends.

Introductory Statement and Significant Points in Progress

Among the significant points in the development of mineral industry education in the United States since 1871, the following deserve particular mention:

1. The number of institutions offering undergraduate instruction has increased, particularly during the years 1890 to 1925; the number of students enrolled in undergraduate study has increased greatly, largely on account of the development of courses in petroleum engineering and increasing interest in metallurgy.

2. The selection of students to be admitted to undergraduate courses has been improved through the advisement, counseling, and guidance programs sponsored by the Engineers' Council for Professional Development.

3. Improved instruction methods and the training of the teaching staff have been promoted by the educational institutions themselves, but with the most effective stimulus and guidance of the American Society for Engineering Education and the Engineers' Council for Professional Development.

4. The accrediting of undergraduate curricula through the Engineers' Council for Professional Development, considering both qualitative and quantitative criteria, has done much to improve standards of instruction in engineering education for the mineral industry.

5. Undergraduate curricula and courses are revised currently to meet the needs of the American mineral industry. New curricula are developed currently to permit specialization in technical fields, particularly in the fourth year of undergraduate work.

6. There has been a strengthening of the fundamental courses in mathematics, physics, mechanics, and chemistry, and at the same time these courses are being adapted to the most recent developments in the industry, including metallurgy, geophysics, petroleum engineering, petroleum technology, ceramics and so forth.

7. There has been continuing improvement in plant and laboratory facilities of the educational institutions.

Manuscript received at the office of the Institute March 10, 1947. Presented before the Mineral Industry Education Division. Issued as TP 2264 as Mining Technology, November 1947.

* Consulting Mining Engineer, Pittsburgh, Pennsylvania.

8. The publication in the United States of textbooks and other technical literature has made it possible to use the time of both teachers and students more effectively and has served to organize a great wealth of technical and scientific information of instructional value. The technical press, the United States Geological Survey, the United States Bureau of Mines, state surveys and mining departments, and the various professional societies and institutions, have been important factors in making technical and scientific information available.

9. The continuing development in the United States of mineral industry education has made it possible for students to receive the highest grade of instruction and there is no occasion for them to go abroad for undergraduate work.

10. Facilities for graduate work and for research have been improved, particularly in metallurgy, mineral dressing, geology, geophysics and petroleum engineering. In recent years there has been greater interest in graduate work and research in mining engineering, and many schools presently are expanding their facilities to accommodate these phases of training in this important branch of mineral industry education.

11. The engineering profession and the mineral industry have shown increasing interest in the education, training and professional development of young men.

In measuring progress in the development of mineral industry education, it is important that the relationship between the changes and expansion in the mineral industry and of technical education be noted, because education for the mineral industry has followed growth and change of the mineral industry. Technical or general engineering education of college grade was started in the United States several decades before education in mining and metallurgy was organized. When the Institute was established in 1871, there were several colleges offering instruction in mining and metallurgy. Columbia School of Mines had been established in 1864 and since that time there has been continuing progress and development in the various phases of education for the mineral industry.

What is "progress" in mineral industry education and how can it be measured? During the 75-year period since the founding of the Institute, there has been remarkable growth in the number of educational institutions teaching mining and allied subjects, the number of curricula offered, the number of faculty and students, the number of graduates per year, the size of the plant facilities and the total annual budgets. The earnings of graduates have increased and much greater interest in this type of education is being shown by the mineral industry, by the federal and state governments, and by philanthropic individuals. All of these matters deserve consideration in estimating progress; but, undoubtedly, all will agree that among the factors by which progress may be measured we should give first place to the quality of the training and instruction, particularly in relation to the needs of the changing mineral industry and of our industrial society.

Detailed discussion of statistics of enrollment and of graduation is unnecessary. Professor William B. Plank has collected and published reliable data currently in *Mining and Metallurgy*. The essential data collected in the sixth biennial survey of mineral technology schools[1] with corresponding data for previous years are given in Table 1, which also shows the trend of distribution by courses. There were 520 graduate students in 1940 to 1941, as compared with 285 in 1934 to 1935.

These figures show the striking enrollment increase in metallurgy, petroleum and natural gas since 1933.

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[1] References are at the end of the paper.

It is interesting to note that of the 1454 who received their first degree in the United States schools in 1940, a total of 1247 took employment in the mineral industry, 100 took employment in other fields of work and 107 enrolled for graduate work.

TABLE 1—*Data Showing Trend of Distribution by Courses*

Courses	1932–1933	1933–1934	1934–1935	1936–1937	1938–1939	1940–1941
Mining.......	1,393	1,161	1,526	1,652	1,892	1,978
Metallurgy...	1,288	937	937	1,630	2,403	2,450
Petroleum and Natural Gas	726	681	1,286	2,466	3,538	2,877
Ceramics.....	522	629	559	738	825	843
Geology......	387	438	524	567	844	792
Fuel Technology........	67	70	77	66	24	44
Special Students.......	60	79	102	71	93	149
Totals....	4,443	3,995	5,011	7,190	9,619	9,133

Reference will be made later to the increase in the number of educational institutions offering credited curricula in the field of the mineral industry.

PROGRESS IN MINERAL INDUSTRY IN UNITED STATES

Any analysis of the progress or of the trends that have been evident must be related to the conditions prevailing in 1871, both in the mining industry and in mining education as it existed at that time.

When the Institute was organized, it was stated that the chief objectives were promoting the arts and sciences connected with (1) the more economical production of useful minerals and metals, and (2) the greater safety and welfare of those employed in these industries.

It was proposed to achieve these objectives by means of meetings for social intercourse, the reading and discussion of professional papers and circulating the information thus obtained among the members and associates by means of publications.

The educational objectives are restated at this time in order that the important part that has been played in mining education by the Institute may be related to the whole field of education and appraised properly. As will be pointed out later, the educational features of the Institute have been emphasized in various ways in addition to the holding of meetings and the publication of technical papers.

A large percentage of the Institute membership of 243 in October 1873 was identified with coal, iron and steel, but the predominating interest was in metal mining. The young men enrolling in the mining schools were drawn there chiefly by the romance of mining. As the mineral lands of the west were explored more thoroughly and the fabulous discoveries of ore deposits, valuable particularly for the precious metals, became less frequent, the situation changed.

The glamour of the gold fields of California, of the Comstock, and of other districts glorified by T. A. Rickard in his volumes, *The History of American Mining* and *The Romance of Mining*, has gone. Instead of glamour we have the problems of engineering and science relating to mass-production, efficiency in recovery in mills and refineries, production and conservation of oil and gas by engineering, the development of new alloys, the manufacture and efficient use of tools and machines in mines, mills, and smelters and the economic application of power in winning low-grade ores and low-priced mineral products.

RECOGNITION OF TECHNICALLY TRAINED MEN

On an occasion like this, it is proper that attention be directed to the part that technically trained men have played in the great advances in mining and the fact that industry now recognizes the importance of technical education and training.

Mr. John Hays Hammond said that the development of the porphyry coppers made

possible the greatly extended use of electricity, because thereafter there was assurance of an abundant supply of copper at a reasonable price. In speaking of the accomplishments of the technical men who initiated the mining of these low-grade coppers, he said:

> They literally had to make something out of what was nothing of economic value before. The development of these great enterprises has changed the lives of the people of whole states in our West and of nations in South America. It is one of the great epic stories of the times; one in which the heroes and leaders are our own professional brethern, and almost without exception of our professional society.

Mr. A. B. Parsons in the Introduction of his volume, *The Porphyry Coppers*, said:

> It is an outstanding example of the advantages of mass-production coupled with the fruits of scientific research and inventive genius. It reflects the pioneer spirit, the initiative, the self-reliance, the independence of thought and action that are traditional with the mining engineer.

Much of the progress made in underground metal mining in the United States during the period from 1905 to 1920 was due to the work of young Americans trained in technical institutions. One of the most important factors in the great progress of that period was the introduction of the high-speed one-man drill replacing the cumbersome two-man drill. To use such drills effectively, it was necessary to have improved drill steel and the young college men of the West cooperated with the manufacturers in applying to drilling practice the most advanced knowledge regarding special alloy steels. This revolution in drilling contributed to improvements in breaking ground and the young mining engineer of that period earned a leading place in metal-mine production technique which had been monopolized previously by non-college men who followed rule-of-thumb practice. From Colorado the use of these drills was extended into the other Rocky Mountain States and later into Missouri and the Lake Superior districts.

About the same time American industry began work on what is now called scientific management, including time-studies in shovelling and other underground operations.[2] The mining engineer has done pioneering work in this field and has made important contributions in improving production.

The block-caving system, first employed successfully many years ago in Michigan iron mines, has been studied in its relations to the character and the strength of the constituent material, the engineering fundamentals, and the forces acting. The extended use of this and other types of mining has been in part because of the intelligent application of sound engineering and knowledge of good practice by qualified and experienced engineers.

The extensive use of machinery in coal mines, particularly in the mechanical loading of coal, created an attractive field for technically trained men, and they have been most active in underground coal mining practice. The day has passed when the traditional pattern of a mining system may be applied without modification to the mining of a coal field. Generally, coal mines are planned to use most effectively the design or type of available equipment best suited to the specific conditions and to take full advantage of power equipment so as to permit mass-production, streamlined transportation and effective ventilation, with men and power being used to produce maximum tonnage safely from concentrated workings.

The Climax Molybdenum Co. has recognized the need for technically trained men in its enterprise. In their opinion, science in mining is as progressive as chemistry, medicine, physics, or any of the other sciences, and that keeping abreast of im-

proved mining methods will, undoubtedly, find many of the present-day underground practices antiquated and obsolete in years hence.

In the iron and steel industry the technologist has established himself on account of many advances made in the last twenty-five years, and particularly since 1930. The productive capacity of blast furnaces, open-hearth furnaces, Bessemer furnaces and electric furnaces, has been increased tremendously through improvements which require technologists in both the design and operation of such equipment. By precise controls it has been possible to improve greatly the quality and reliability of standard products, to produce a great variety of alloys and to use the plant with greater efficiency and lower operating costs. There has been a great improvement in the use of fuels and the effective application of much of what was waste heat.

In 1928 the Institute published a volume on flotation and in the preface it was stated that flotation was the most important development in recovery of minerals from ores in the present century. "No other process has effected such a great change in metallurgy in so short a time." The technologist has played a most important role in the development and application of this practice.

The oil and gas industry is probably on as advanced scientific and engineering basis as any of the other mineral industries. This has come about by parallel progress in developing the scientific principles pertaining to the occurrence and recovery of oil and gas, to petroleum engineering schools for training technical men, and to an increasing appreciation on the part of management of the worth of science and engineering, and finally to an aroused public which has demanded conservation and the elimination of the flagrant wastes which plagued the industry in the early days.

The field for the technical graduate has grown rapidly through the years because of his ability to adapt himself to new situations, to apply the best tools available at the time and to develop more efficient tools and methods as rapidly as conditions permit.

What has been said in regard to the recognition given the engineer in production, dressing and processing of ores and solid fuels in the steel and petroleum industries, applies equally well to other branches of the industry including metallurgy, geology and ceramics.

In summarizing the changes in the mining industry that have shown the need of the technically trained man, the following may be noted:

1. Among the early advances which gave the college man a chance were the smelting of complex ores, improved ore-dressing practice, the introduction of such processes as cyaniding and chlorination and the necessity for the increased use of power and power machinery, both underground and on the surface.

2. Later advances that contributed were the installation of the one-man drill and improved drill-steel, the efficient service of the geologist underground in planning and directing exploration and development of ore deposits, the inauguration of open-pit mining and mass-production methods, improved iron and steel metallurgy and the safety program for coal and ore mining.

3. More recent advances that have been important include: the greater use of machinery underground, the application of scientific management to production problems, the more extensive mining of low-grade ores, the engineering approach to mining methods, greater advances in the technology of iron and steel manufacture, the production of alloys and light metals, the development of petroleum engineering, the extensive application of geophysical methods of prospecting, the greatly increased use of industrial minerals with engineering and science applied to production and processing, the great advances in mineral dressing, including the recovery of

byproducts, the greater application of power and machinery to reduce labor costs, the extended research in all phases of production and processing of minerals and the general recognition that thorough training in engineering and science is invaluable in developing men for leadership in all branches of the mineral industry.

Activities of the Institute in Mineral Industry Education

The American Institute of Mining and Metallurgical Engineers was organized to make possible and to stimulate the exchange of information relating to the mineral industry among engineers, metallurgists, and others actively engaged in the industry and to promote the interests of members of the profession. However, it was soon realized that one of the most important things that could be accomplished would be in assisting young engineers to improve themselves and to qualify for more responsible positions in the industry. As the years went by the importance of this phase of service to the industry received greater recognition and more attention of the Institute was directed to the work being done in the mining schools and technical colleges. Prior to 1929 there were no special student membership grades, but there were affiliated student societies at a number of the mining schools. In 1920, there were 32 such affiliated student societies.

The members of the teaching staff of the various mining schools have always played an important part in the activities of the Institute, but for many years there was no coordinated effort on the part of the institutions giving mining courses to improve the curricula because each established its own standards and planned curricula according to the teaching staff and facilities available. However, some of the western institutions made it a practice for a number of years to secure the younger members of the teaching staff from among the recent graduates of the eastern and older institutions. Through this practice there was a wholesome and stimulating exchange of ideas and, in some instances, a definite improvement in scholastic standards.

The activities of members of the Institute along educational lines have been notable through the years. A number of the institutions giving instruction in mining and related subjects have received from mining men and mining corporations substantial gifts, in the form of buildings, endowments of chairs, or as funds to finance research, scholarships, fellowships, or student loans. Reference will be made later to the funds administered by the Institute for educational purposes.

Various of the divisions and local sections have well-administered programs for awarding scholarships, fellowships, and prizes. These provide educational opportunities for worthy young men and stimulate interest in the work of the Institute.

Another substantial organization affiliated with the Institute committed to a continuing and substantial educational program is the Woman's Auxiliary which has established scholarships and loan funds for outstanding young men desiring an education for the mineral industry. This work has been well-supported and splendidly administered through the years and represents a distinguished contribution in the educational field.

At the present time the student-aid program of the Auxiliary is in the form of "Scholarship Loans" as the beneficiary is expected to repay 50 pct of the total sum granted. The program began in 1922 and a total of 155 scholarship loans have been made, amounting to more than $95,000. Among the men and women who have been holders of these scholarships 131 have graduated. In the peak year the Auxiliary had 29 young men in college.

In addition to the publication of technical papers, volumes of proceedings and special volumes, the organized work of the

Institute in the field of education has taken the form of work of standing committees to improve the relations with student chapters, to assist in securing speakers for student meetings, and to provide funds for student prizes for technical papers.

On April 24, 1924, the Board of Directors appointed a Committee on Engineering Education and eight years later this work was organized as a Division of the Institute. On March 16, 1934, the Directors appointed the first Committee on Student Relations. The first work of this Committee was to take steps to improve relationships between students and Local Sections, and the Institute at large.

It is fitting that some mention be made of the splendid work that this Division has done since its organization. Not only has it provided the opportunity for the exchange of ideas relating to methods of instruction, but it has served in a splendid manner in originating programs through which the Institute and the industry could improve mineral industry education and research and coordinate programs.

Early Development of Engineering and Mineral Industry Education

Having noted the organized work of the Institute in the field of mining education, the early development of engineering and mineral industry education will be reviewed briefly.

As a number of engineering schools were established in the United States before any mining school was opened, it is appropriate to note the work of these early engineering schools as they influenced the development of the later mining curricula. The first engineering schools in the United States were organized during the period 1810 to 1820 and the curricula showed the influence of the French system. The purpose of the founder of Rensselaer Polytechnic Institute (Troy, New York) was the training of teachers of natural science. Instruction in engineering came slowly, the early courses including land surveying, measurements of the flow of water in rivers and aqueducts, hydrostatics and hydrodynamics and calculations upon the application of water power and steam. The degree of Civil Engineer was awarded for the first time in 1835 with a class of four.

President Greene of Rensselaer visited leading institutions in Europe in 1846 and analyzed their curricula in order to reorganize the course of instruction at Rensselaer. Dr. Wickenden says:[3]

> Without doubt Greene was the first man in America to submit the problems of education for the technical professions to thorough investigation and analysis. In essential matters his conclusions remain sound today. What he visualized was an educational discipline complete in itself, not narrowly utilitarian but adapted to the complete realization of true educational culture.

The period leading up to 1860 was one of pioneering in engineering education. Foundations were being laid by Horace Mann and others for a comprehensive system of public education, and when the Morrill Land Grant Act was passed in 1862, it showed the desire of public leaders to provide training for farmers and mechanics. This Act put the resources of the Federal Government behind the popular movement for "the application of science to the common purposes of life." It was stated that the impetus of the Morrill Act was felt by private as well as public institutions. In the space of a single decade, from 1862 to 1872, the number of engineering schools increased from 6 to 70.

The Morrill Act opened the way for the rapid expansion of engineering education through the north and the west, as it provided for the support of state colleges of agriculture and mechanic arts. By 1870 twelve states had accepted the provisions of the Morrill Act.

The coal and iron industries in the Middle Atlantic States, and metal mining in the west, developed rapidly following 1850.

American mining industry had been dependent on European schools of mining and technology for the training of mining engineers and metallurgists. When Columbia School of Mines was opened in 1864, the curricula followed somewhat that of the Ecole des Mines of Paris. Most of the American schools established from 1865 to 1880 followed in general the plan of Columbia.

In speaking of the development of engineering generally, Dr. Wickenden says:

> After 1870 came a period of expansion and ramification, based on the models already created; engineers of distinction took an increasing leadership in education; an American literature of engineering began to develop through the authorship of leading professors; the engineering profession took on solidarity and began to influence the scheme of education; and an extraordinary expansion of industry created a wide field for mining, metallurgical, and mechanical engineers.

The 1870 to 1885 period has been referred to as the great formation period in American engineering education. The Centennial Exposition in 1876 did much to popularize and stimulate technical education. During this period the collegiate type of curriculum with its base of science, mathematics, languages and social studies became firmly established.

A noteworthy feature was the great development of laboratory methods of teaching, a field where American leadership has been especially marked. Laboratories were established at the early mining schools. Methods of shop instruction were developed by mechanical engineering departments.

In 1893 professional consciousness had not been developed among engineers. There was still the conflict and discussion as to the merits of classical and so-called practical education.

Laboratory instruction continued to expand and improve, and by 1900 American laboratory installations and methods for teaching of engineering were not surpassed and not often equalled in any other part of the world.

There was an improvement in standards for admission to engineering colleges and improvement in collegiate instruction in the sciences. Engineering literature in the United States increased rapidly and engineering education turned definitely from European models and developed methods and practices better suited to the requirements of our type of industrial society.

It is not the plan to review many historical details of mineral industry education, as Dr. T. T. Read has presented them in a most complete and scholarly manner. Following the comprehensive chapters on *The Beginnings of Mineral Industry Education* and *American Beginnings*, he presented interesting data on the status of the European schools of mines, including a list of the prominent Americans who attended these institutions from 1850 to 1865.

As a result of the vision of several of the Americans who had attended European mining schools, the School of Mines of Columbia College, New York City, was organized and opened officially on November 15, 1864. While mining instruction had been given at an earlier date at several other institutions, it is generally accepted that Columbia was the first to establish and maintain a course of instruction in mining subjects leading to a degree. Of the total number of mineral industry degrees given in the United States up to 1890, probably more than half were granted by the Columbia School of Mines.

In 1871, when the Institute was formed, there were eight American colleges granting degrees in mining. In 1876 the newly-elected president of the Institute, Mr. A. S. Hewitt, presented a paper entitled, *A Century of Mining and Metallurgy in the United States*.[4] After pointing out that the United States could then more than supply its own requirements of coal, iron, gold, silver, lead, zinc and copper, he named two agencies

which had been particularly instrumental in the "steady social and economical improvement" of the mining and metallurgical industry. First, he mentioned the dissemination of knowledge of mineral resources throughout the country by such reports as those of the United States Commissioner of Mining Statistics and technical periodicals. Next in order, he ranked the influence of technical schools, and said:

> The number of these has increased rapidly during the past ten years; and I venture to say that many of them compare favorably, in theoretical instruction at least, and several of them in the apparatus of instruction, with the famous schools of the old world. The Massachusetts Institute of Technology, the School of Mines of Columbia College, the Sheffield Scientific School of Yale College, the Stevens Institute of Technology, the Pardee Scientific Department of Lafayette College, the excellent school at Rutgers College, the new Scientific Department of the College of New Jersey, the School of Mining and Metallurgy of Lehigh University, the School of Mining and Practical Geology of Harvard University, the School of Mines of Michigan University, the Scientific Department of the University of Pennsylvania, the Missouri School of Mines and Metallurgy, the Polytechnic Department of Washington University, and the similar department of the University of California, and perhaps some others which I have omitted to name—this is a list of schools for instruction in the sciences involved in mining and metallurgical practice, of which we need not be ashamed. What our schools undoubtedly need, is a more intimate relation with practice.

Dr. Read has dealt with this early history in an admirable manner, and the only reason this quotation is presented is to emphasize the interest in mining education shown by the Institute 70 years ago, and also to indicate that while a number of the same schools are now giving curricula for mineral industry education, some of these listed no longer give instruction in mining subjects.

During the decades from 1890 to 1910, there was a notable development of state schools and increased interest in mineral industry curricula. The registration at the University of California increased threefold from 1893 to 1903. By 1896 mining courses had been organized at not less than 24 institutions, in addition to the eight granting degrees when the Institute was organized. In the 25-year period, from 1896 to 1921, a number of other institutions added curricula in mining, metallurgy, geology and petroleum, and in the 25-year period following 1921, the new curricula were largely in metallurgy and petroleum.

The Engineers' Council for Professional Development in its last report of accredited institutions and curricula, listed 26 accredited in mining (and geological engineering) and 27 in metallurgical engineering.

A review of the history of mining education during the period 1871 to 1896 must emphasize the importance played by a few outstanding institutions, the work of a number of distinguished scholars and teachers and the laying of foundations which have characterized much of the educational program of later years. Evidence of this is given in papers published by leading educators of that period. Professor S. B. Christy gave a paper in 1893 on *The Growth of American Mining Schools and Their Relation to the Mining Industry*;[5] H. O. Hofman presented a paper on *The Equipment of Mining and Metallurgical Laboratories* in 1895; and H. S. Munroe gave a paper on *A Summer School of Practical Mining* in 1880. Dr. R. H. Richards presented a paper on *The Mining and Metallurgical Laboratories of the Massachusetts Institute of Technology* in 1873, and one on *American Mining Schools* in 1886.

A study of these papers indicates that the pioneer educators of those days were dealing with some of the same teaching problems that confront educators of today. Dr. Read has pointed out the early objectives

and standards, the breadth of the curricula offered, the emphasis on the fundamentals of science and engineering and the early development of laboratory courses and summer field work of various kinds. Not the least of the factors contributing to the progress during the first quarter century was the inspiring leadership of men in the faculties of the various institutions. These men gave devoted service and as a result of their splendid work there was little occasion for Americans to go to Europe for undergraduate work for the mineral industries, as had been the custom before 1865.

Important Factors in Development of Engineering Education

One of the most influential factors in the development of engineering education in the United States has been the work of the American Society for Engineering Education (until July 1946, the Society for Promotion of Engineering Education). Founded in 1893, it has played an important part, not only in scheduling meetings where college teachers may exchange ideas by the presentation of papers, but also by constructive programs planned to give the best possible training, to keep pace with progress in industry and to anticipate the future needs in the educational field.

The Study of Engineering Education by Dr. C. R. Mann, under the auspices of The Carnegie Foundation for the Advancement of Teaching, was the first of a series of stimulating processes in the field of technical education initiated generally by the great engineering societies. The first steps toward starting this study were taken in 1907, but the final report of the investigation was not issued until 1918. Among other things, this report called attention to regulation of admission, the failure of a large number of those admitted to complete the course, and the desirability of better selection of those enrolled in the colleges, better organization of curricula and controlling the content of courses so that departments would not be the sole arbiters of the content of courses, broader training in fundamentals forming the "common core" of every curriculum, and consideration of improvement of character, resourcefulness, judgment, efficiency, understanding of men, and technique.

In 1922 a Development Committee of the American Society of Engineering Education was appointed to consider "what the Society can do in a comprehensive way to develop, broaden, and enrich engineering education." This Committee recommended that a comprehensive survey be made and a complete investigation of engineering was planned. In 1923 a five-year program was started, the first three years being financed by the Carnegie Foundation for the Advancement of Teaching. The concluding portion of the work was supported by various engineering societies, industries, and individuals.

The work of this investigation, done in 1923 to 1929, was published in one volume in 1930, and in 1934 a second volume was issued, including a Study of Technical Institutes (noncollegiate). Dean A. A. Potter said that the investigation of engineering education in 1923 to 1929 "gave the greatest and most beneficial impetus to engineering education which it had received throughout its entire history."

As a result of this investigation, definite conclusions were reached and several steps were taken to make these effective:

1. Engineering education cannot be cast in a fixed mold. Educators must be alert to meet the conditions created by fruitful scientific research, expanding industry and the enlarging dependence of modern life upon engineering activities.

2. The need for continuing surveys of engineering education was recognized.

3. Summer schools for engineering teachers were organized.

4. It was believed that a formal general agency, such as an Engineering Education Foundation, would be of outstanding value in developing continuously engineering education as a potent factor in natural progress.

The program of the Engineers' Council for Professional Development, organized in 1932, includes assisting in the improvement of engineering education and formulating "criteria for colleges of engineering which will insure to their graduates a sound educational background for practicing the engineering profession."

The procedure of accrediting engineering curricula, beginning in 1935, has included schools of the mineral industry and, while the Engineers' Council for Professional Development has no authority to impose any restrictions or standardization upon engineering colleges, it has had a very wholesome effect.

The basis for accrediting undergraduate curricula of engineering colleges includes both qualitative and quantitative criteria. These are published in the annual reports of the Engineers' Council for Professional Development and have been accepted as being comprehensive, fair, and conducive to superior instruction and progressive policy in educational matters.

Upon the request of the recognized officials of an institution, the accrediting committee has given informal advice on matters such as changes in educational policy, in organization, in curricula, personnel, and physical facilities. This has proved an important phase of the committee's work in advancing engineering education.[6]

The Engineers' Council for Professional Development has recognized six major curricula, including metallurgical and mining engineering, and "such other curricula as are warranted by the educational and industrial conditions pertaining to them."

An examination of the list of accredited curricula, as of Sept. 30, 1944, shows the following: (includes 48 educational institutions)

Ceramic Engineering (including technical options)	10
Fuel Technology	1
Metallurgical Engineering	
Accredited curricula	30
Options as part of other accredited curricula	7
Mining Engineering	29
Petroleum Engineering	
Accredited curricula	13
Options as part of other accredited curricula	6
Geological Engineering	
Accredited curricula	3
Geological option	1

It is recognized generally that the work done by the Engineers' Council for Professional Development in the accrediting of engineering curricula has been outstanding and of great value to American industry, as well as to engineering education.

Progress in Undergraduate Instruction in Mineral Industry Education

As has been noted previously, any comparison of instruction must relate the later period to some of the earlier periods; comparisons should consider the same qualitative and quantitative criteria as are now used by the Engineers' Council for Professional Development accrediting committees. The facilities for instruction, including available textbooks, technical literature and so forth, must also be considered. The relationship of industry to technical and scientific education has been noted, as well as a number of other factors. Progress in education has followed progress in the industry.

Notable improvements have been made in the textbooks and technical literature available in all fields. It is unnecessary to list the available technical literature on such dates as 1871, 1896, and 1921, but it should be noted that prior to 1900 there were few American textbooks in the field of mineral industry. The teaching was largely by lectures and assigned readings in technical journals and proceedings of technical societies. The availability of well-organized, up-to-date textbooks marked a tremendous advance in the education work.

The early curricula included mining, metallurgy and geology. As new methods of treating ores were developed, the instruction in metallurgy became more specialized and options or curricula in metallurgy were offered. This practice of recognizing special needs for instruction to qualify men for industry has been followed, notably in the development of curricula or options in ceramics (1896), petroleum (1912), geophysics (1926), mineral dressing (1934), geological engineering (1922) and fuel technology (1932).

There has been much discussion through the years in regard to four- or five-year courses in engineering. A detailed review of the experiences of various institutions that have inaugurated five-year courses is not appropriate at this time, but the concise statements of Dr. Read in regard to the experience of several institutions is pertinent, as follows:

> At Harvard after 1934 a bachelor's degree was required for admission to the Engineering Schools and at Stanford this had been adopted as early as 1925. The Dean of Engineering at Harvard later characterized its policy of dropping the four-year engineering curricula as a 'costly mistake.' The effect on mineral industry registration does not seem to have been as immediately serious at Stanford as at Columbia, but was eventually. At Columbia the effects were so severe on the whole Engineering School, that curricula requirements were progressively so modified that an adequately prepared student can now obtain a B.S. degree in an engineering field in four years and the engineer degree in five. Even with this liberalizing, the mineral industry undergraduate registration has never recovered from the blow dealt it and the major interest of the mining and metallurgical department has become graduate work.[7]

Detailed comment on the historical development of the courses is unnecessary as the entire field has been covered by Dr. Read, but some of the most important developments and trends will be noted.

Mining

While the amount of time given to mining courses in undergraduate curricula is somewhat the same throughout the schools of mineral industry, and while there has been no substantial change through the years, there has been marked change in the content of the courses now being given, in contrast with those given 25 and 50 years ago. There has been considerable discussion regarding mining instruction and some criticism of the content of the courses, but in fairness to the teaching staff at the various schools it is very evident that progress has been made. This applies both to metal mining and coal mining. Commendable effort has been made to keep abreast of the industry and to cooperate in research relating to drilling, explosives, underground machinery of all types, the application of power, behavior of roof and mineral being mined, ventilation, haulage, drainage and safety engineering.

Improved underground mining methods and open-pit or strip mining have revolutionized production. Mines are planned to make effective use of available equipment, or machinery is designed to meet new and changing conditions. The college-trained man has been an important factor in the progress that has been made and the mining schools have adapted their instruction to the changing practice. Conventional courses in plant design and so forth, have been improved to meet current requirements of the industry.

Metallurgy

Substantial changes have occurred in the teaching of metallurgy and in the content of the courses and curricula.

In discussing the content of metallurgical engineering curricula in the United States in 1940, Professor H. L. Walker pointed out[8] that a large percentage of eastern schools require subjects in the following group: electrometallurgy, engineering mate-

rial, metallography, physical chemistry, physical metallurgy, thermodynamics and X-ray. A large percentage of the western schools require such subjects as the following: design, fire assaying, hydraulics, metallurgical analysis, mineralogy and geology, mining, ore-dressing and surveying.

Dr. Read has reviewed the great advances in teaching metallurgy, including the introduction of metallography, process metallurgy and metallurgical calculations during the period 1897 to 1907. In 1908 Dr. Bradley Stoughton published his *Metallurgy of Iron and Steel,* emphasizing the importance of instruction in physical metallurgy. Dr. Read points out that starting as but little more than a description of an ancient art in the sixties, metallurgical teaching had by the 1900's advanced, along with the enormous advancement in the art itself, to where principles as well as practice were being taught.

There has been a substantial change in the content of the courses in the last 25 years. Undergraduate instruction in metallurgy in most institutions comprises not only a study of the fundamental principles but a thorough examination of the processes being employed in industry to produce metals and alloys having special chemical and physical properties. The courses of the sophomore and junior years have been reorganized to supply the training prerequisite to the advanced courses, which may be elected according to the curricula or options.

Following the early work of Sauveur, Howe, and others, the teaching of metallography soon came to play an important part in instruction in metallurgy and in later years X-ray has been added in the instruction in physical metallurgy. Largely because of the exacting requirements of the automotive industry, great strides were made in research in iron and steel and alloys. This work has resulted in improved instruction in undergraduate as well as graduate work. The greatly extended use of light metals and the development of numerous alloys has resulted in specialization within the metallurgy curriculum and the development of a number of new courses.

The most significant trends have been greater emphasis on physical metallurgy, metallography, X-ray and so on, and a less emphasis on laboratory work in qualitative analysis, quantitative analysis and fire assaying.

Mineral Dressing

Among the subjects first taught in American mining schools was ore-dressing. The early work of Dr. R. H. Richards at Massachusetts Institute of Technology is referred to by Dr. Read as "of controlling influence both in the practice and teaching of the art in the United States." It has been said that Dr. Richards developed a German art into an American science.

Ore-dressing and coal washing were formerly considered as part of the mining courses, and were generally taught by men in the mining department, but in more recent years this work has been developed greatly. In certain institutions a separate curriculum has been organized to replace the ore-dressing option formerly given; in other institutions ore-dressing is now taught in the metallurgy department.

Dr. Read has discussed this subject in his chapter on *Special Curricula Other than Petroleum.* He pointed out that since 1900, when froth-flotation became of greater importance, the dividing line between metallurgy and ore-dressing became less distinct. In 1934 to 1935, Columbia announced a curriculum in mineral dressing, and in 1940 to 1941 this curriculum was revised. It was announced that the aim of this curriculum was to train its graduates in the methods used or available for dressing the crude minerals comprising the crust of the earth into the primary derivatives which constitute consumer materials.

Great advances have been made in a number of phases of mineral dressing in part resulting from the mass-production of lower-grade mineral materials. This has required improved recovery and lower costs. Fundamental research in laboratories, pioneering work in pilot plants, and continuing checks of raw materials, products and refuse or tailings have required a large force of skilled men and technical supervision. The changing industry has required continuing change in the technological courses of instruction.

Geology

Geology has been taught in the mining schools as both a pure science and an applied science. More and more emphasis has been placed on those subjects generally referred to as in the economic field, and in certain instances the broader aspects have been emphasized. The courses in mining geology have been adapted to the changing needs of industry.

In his Presidential Address presented before the Society of Economic Geologists in 1945, Mr. John M. Boutwell gave his ideas on the requisites in education and in technical training for making a first-class economic geologist. In his opinion the best preparation should embrace a broad general college education, intensive graduate specialized study in advanced geology and allied subjects and apprenticeship in actually doing in association with experienced successful economic geologists.

Mr. Boutwell pointed out that up to less than 50 years ago, geological work in connection with mining operations was generally regarded as mining engineering. Following the pioneering studies of Irving, Emmons, Van Hise, Lindgren, Penrose and Smyth, the geological study of mineral deposits underwent marked expansion. The United States Geological Survey made monographic geologic studies of many of the most important mineral districts.

During this period the growing use of scientific aids, such as petrographic and chemical study of rocks, physical chemistry, examination of polished ores in reflected light, experimental reproduction of conditions of high temperature and pressure, and improved technique in laboratory and field, were yielding significant contributions to our knowledge of mineral deposits.[9]

Beginning about the year 1900, large mining companies began to organize geological departments, realizing the importance of systematic mapping of underground geology for the guidance of development and mining operations. Among the first was the Anaconda Copper Mining Co. and by 1928 the geological department of this company was employing 15 geologists, 44 engineers, 67 samplers and 11 draftsmen. The very interesting development of the geological departments of other companies has been described by Mr. Boutwell, including those of Oliver Iron Mining Co., Cleveland Cliffs Iron Co., Greene Cananea Copper Co., Phelps Dodge Corp., Pickands, Mather and Co., M. A. Hanna Co., Cerro de Pasco Copper Corp., and Calumet and Hecla Consolidated Copper Co. He stated that these pioneering geological departments afforded such valuable results that one company after another sought the advantages of using geology in mining operations, not only in metallic mineral deposits, but also in nonmetallic deposits, notably petroleum, as well.

Dr. Read has discussed geological engineering which is not at all identical with mining geology, the former dealing particularly with the geological problems of civil engineering encountered in the design and construction of such works as sub-aqueous tunnels, water-supply conduits, and bridge and dam foundations. Courses in geological engineering have been accredited by the Engineers' Council for Professional Development at the Colorado School of Mines, Montana School of Mines and New Mexico

School of Mines. At the University of Idaho there is an option in geology in the mining engineering curriculum.

Reference will be made later to the educational features of the work of the United States Geological Survey.

Geophysics

In 1929 the American Institute of Mining and Metallurgical Engineers published a volume on *Geophysical Prospecting*. In the preface, Dr. Donald H. McLaughlin, who was Chairman of the Committee on Geophysical Prospecting, said:

The sudden realization of the possibilities of practical service from geophysical methods of prospecting for ores or other valuable bodies has recently aroused very general interest on the part of mining and oil men, geologists and physicists. Although few of the methods now actually practiced are entirely new, relatively little use was made of them until the last few years."

Undergraduate instruction in this field has been offered at a number of educational institutions. Dr. Read reviewed the development in the United States and pointed out that the Colorado School of Mines (Golden, Colorado) established a separate department of geophysics in 1926. In 1930, M. King Hubbert in a paper entitled, *The Place of Geophysics in a Department of Geology*,[10] gave impetus to the work in geophysics and opened the door for discussion of the educational backwardness of the whole field of instruction in earth science. The American Institute of Mining and Metallurgical Engineers has published volumes on geophysical prospecting in 1929, 1932 and 1934, and on geophysics in 1940 and 1945.

The Geophysics Education Committee of this Division submitted a report in February 1943, entitled, *The Professional Training of Geophysicists*. Undoubtedly, this subject will be discussed by other speakers on this program. A number of organizations have participated in the discussion of education in geophysics and earth science, including the Geological Society of America, the American Association of Petroleum Geologists and the Association of Exploration Geophysicists. It should be noted that great progress has been made in instruction in this field, particularly during the last 20 years.

Petroleum

The development of instruction for the petroleum industry has been interesting and inspiring. It is thoroughly American and represents probably as well as any other branch of engineering education, the intelligent effort of the universities and colleges to meet the needs of a giant industry that grew by leaps and bounds.

The early history of education in petroleum engineering, beginning with the University of Pittsburgh, has been recorded by Dr. Read. Details of the content of curricula and courses need not be discussed, but it should be noted that the Educational Section of the Petroleum Division of the Institute made a comparison of specialized curricula in 1943. As a result of the survey, the chairman of the committee, Professor H. H. Power, stressed:

1. The general objectives of the American Society for Engineering Education in engineering education.

2. The "fundamental" approach in engineering.

3. The engineering problem method in the teaching of specialized courses. This constituted an abrupt change from the prior "descriptive approach" and was accepted broadly by other institutions and industry alike.

4. The role of other supporting courses, especially English and Economics.

It was recommended that the "fundamental" approach is necessary in the undergraduate curricula—the graduate years can be devoted to the highly specialized courses

such as secondary recovery, pressure maintenance, and thermodynamics.

In commenting on the changing concepts caused by the rapid changes in industry, Professor L. C. Uren said[11] institutions that planned their petroleum engineering curricula a decade or two ago now find that the petroleum industry has become vastly more technical in its requirements. New methods and techniques must be given consideration in the curriculum and some of these require changing emphasis in the fundamental preparatory subjects. He said that college and university faculties cannot complacently assume that curricula in this field which were considered appropriate 10 or 20 years ago still reflect the needs of the industry that they were designed to serve. Professor Uren also raised the question whether academic authorities could anticipate the trends of industry and equip their graduates with what they will need in order to attain leadership 10 or 20 years after graduation, but took the position that there was no excuse for permitting curricula to fall behind the current needs of industry.

Splendid graduate and research work has been done at several of the institutions and, undoubtedly, these matters will be considered by following speakers.

Ceramics

The first courses and curricula in ceramic engineering were organized in 1894 at Ohio State University, the original curriculum requiring two year's work. Chemistry was stressed and engineering courses were included later in the four-year curriculum. It should be noted that the teaching work done at Ohio by Professor Orton was truly pioneering and was the beginning of work toward the intelligent use of nonmetallic minerals.

Ceramic engineering is now taught at a number of institutions and effective graduate work and research has been in progress for a number of years at most of these institutions. The courses in ceramics are planned to train men for the following industries: refractory, glass, cement, structural clay products, enameled metal, lime and plaster, insulating material, and abrasives.

The ceramics curricula include a broad basis of fundamental engineering and science with specialized courses in fundamental processes and the design and operation of ceramic equipment and plants. Because of the fact that many of the plants are small, it is often necessary for the ceramic engineer to be responsible for all the technical work involved in the operation and to conduct research as well. Therefore, it is necessary for the undergraduate course to be most thorough in fundamentals and to include sufficient training in operation to permit the young graduate to undertake diversified work in operating plants.

In connection with the New York State College of Ceramics, there was organized in 1936 the New York State Ceramic Experiment Station. Ceramic research is conducted by five full-time men in addition to work by members of the teaching faculty and the advanced students.

Fuel Technology

Courses and curricula in fuel technology were established in recent years. At Pennsylvania State College an option in fuel technology was part of the metallurgy curriculum in 1930 to 1931; the courses offered to undergraduate students including fuel testing and calorimetry, coking, classification of coals, liquid and gaseous fuels, carbonization and processing of coals, combustion and utilization of solid, liquid and gaseous fuels.

In 1932 a separate curriculum was established and with minor changes the courses offered were the same as those offered under the metallurgy option. The curriculum was created because of the importance of the fuel-producing and fuel-

consuming industries in Pennsylvania and the growing need for men trained in the technology of fuels. Graduate instruction and training in research methods were offered.[12]

Colorado School of Mines offered a fuel option in the third and fourth years of its mining engineering curricula.

Economics and Related Subjects

There has been much discussion in regard to the teaching of economics and considerable progress has been made, particularly at those institutions where there are members of the faculty who have taken fundamental courses in economic theory and have had some experience in technology in the mineral industry, as well as sufficient acquaintance with the business world to formulate a desirable course of instruction. Such a course should give the engineering student a grasp of fundamental principles of economics and the desire to continue his reading and study after graduation. The student should be impressed with the idea that he must grow in his understanding and knowledge of general economic problems, and particularly problems dealing with labor and taxation in the mineral industry. This work has been done well at several institutions.

In a number of the colleges it has been the practice to include in some one of the engineering courses a rigorous training in analysis of cost data, the preparation of engineering estimates and the study of the economic features of construction, installations, operations and so on. Industrial management, time-and-motion study and so-called scientific management are given appropriate consideration in some curricula. However, in most of the schools instruction along these lines is included in a broad and comprehensive course in mine or industrial management, in which many of the economic problems connected with the mineral industry, such as labor, safety, organization and efficiency of operations, mining law, and so on are discussed. This subject has been noted by Dr. Read in connection with *Modifications in Courses at Columbia*, and reference is made to that part of the discussion dealing with management. He said:

> The time allotted to consideration of problems of mine management needed to be increased, and the subject matter needed to be changed so as to deal more directly and effectively with matters likely to come within the range of experience of a young engineer shortly after graduation. Mineral economics was introduced to permit a broad treatment of the economic problems of the mineral industry.

Humanistic-social

There has been continuing discussion through the years in regard to the content of the various curricula and it may be well to note that some of the arguments in favor of a humanistic-social group are not new. The course of study followed by the writer 50 years ago, and leading to a bachelor degree in mining engineering, included two years of English, two years of foreign language, one year of European history, and courses in the senior year in economics, political science and constitutional law.

In his splendid paper presented several years ago at Houston, Professor Power reviewed the program for petroleum engineering education and summarized the ideals and objectives of engineering education as follows:[13]

> Engineering schools have a major responsibility to the public, to industry, and to the professions they serve. The specific purposes of formal education include training in the basic sciences, the engineering-problem method, the rudimentary development of technical skills, an appreciation of values and costs, and an understanding of the art of engineering as distinguished from its science. Other purposes implied include the ability to read, write, and speak the English language effectively, an understanding of social and human relationships, a knowledge of the duties of citizenship,

a broad appreciation of cultural interests, and an indoctrination of professional standards and relations. Throughout these educational processes is the development in the student of habits of accuracy, thoroughness, powers of analysis, creative ability and integrity with respect to all phases of his work.

A report of the American Society for Engineering Education has detailed the objectives of the scientific-technological and the humanistic-social divisions of the program in engineering education and it is not necessary to review them at this time.

Laboratory Work, Summer Schools and Summer Employment

The early volumes of the Institute record the fact that the curricula of the first mining schools emphasized both laboratory work and summer schools for training in surveying, geology, mining, metallurgy and ore-dressing. The Columbia School of Mines held its first school of practical mining in July and August 1877, at Drifton, Pennsylvania. The following year the school was held at Mineville, New York, and in 1879 at the Atlantic copper mine in the Lake Superior District. Detailed studies of mining operations were made by the students who were organized in small squads.

In recent years even greater emphasis has been given to this part of the undergraduate work, some of the institutions requiring the undergraduate student to spend at least one summer as an employee in the mineral industry at the same character of work in which he proposes to specialize. This serves to acquaint the student with some of the practical applications of his classroom work, gives him some idea of operating and production technique and acquaints him with the organization of the working force in industry.

In a number of institutions this summer work is supervised, comprehensive reports are required and college credit is given. In some instances, the summer work gives the student an opportunity to secure data which may be the basis for his undergraduate thesis, where a thesis is required of candidates for degrees.

In connection with the educational features of the work of the United States Geological Survey, reference is made to opportunities for summer work that have been open on United States Geological Survey field-parties.

The cooperative system in technical education attracted attention in American industry about 1906 when Dean H. Schneider started work of this character at the University of Cincinnati. In general, this system has not found favor in the mineral industry, but, undoubtedly, it might be developed in connection with some of the metallurgical industries. In certain instances programs have been considered in connection with mining operations, but have not been made operative.

Instead, it has been the practice for students enrolled in schools adjacent to mines, mills and smelters to work several shifts each week, usually at the weekend. As previously noted, the more common practice has been for the student to spend the entire summer vacation working in the mineral industry, primarily to acquaint himself with operating practice and also to earn money to defray the expenses of his technical education.

Graduate Instruction and Research

The chief emphasis in mineral industry education from its inception has been quite properly on undergraduate study. An authority on graduate work in engineering has advised that there were practically no graduate students in any colleges of engineering prior to 1900. The investigations of the Engineers' Council for Professional Development and of committees of engineering societies have reported from time to time on courses for four-year and five-year undergraduate curricula and

plans for graduate study. Some educational institutions find it desirable to include certain subjects in undergraduate curricula and to group related special subjects in options, thus providing sufficient specialization to meet current requirements. Some of the large corporations which employ a number of graduates each year are prepared to give training in some of the special technical work, and suggest that all the time available in the four-year curricula be given to fundamental science and engineering.

Graduate work leading to advanced degrees is offered at a number of schools of mineral industry, primarily in metallurgy, ore dressing and geology. There has been less opportunity for graduate work in mining, but at the present time such work is being offered at several institutions.

Graduate courses in metallurgy have been offered by a number of institutions depending on the field in which the advanced work is being done. One of the well-equipped institutions stated that the nature and extent of the courses offered depends on the desires and capacity of the student and upon the laboratory equipment available for the purpose. The wide range of subjects from which elections may be made is indicated by the following list in recent catalogs: Advanced hydrometallurgy and electrolysis, advanced metallography, metallurgy and use of the rare metals, applied spectroscopy, advanced non-ferrous production metallurgy, advanced X-ray analyses and advanced mechanical metallurgy.

An examination of college catalogs and correspondence with the mining departments of several of the leading mining schools indicate that graduate work has been carried on or offered in various mining subjects. Among the subjects noted are; (1) mine ventilation and dust technology, (2) mine drainage, pumping, and underground-water control, (3) mine design for special underground problems, (4) research in connection with explosives and blasting, (5) economics of machinery for underground and surface mining, and (6) economics of underground haulage in relation to conveyors, electric and Diesel locomotives.

Valuable research work has been done in recording existing mining practices and in pointing out more extended applications. Much valuable information has been collected through field studies and summaries of these studies have been of great service. In experimental mines or laboratories tests have been made to determine behavior of explosives, coal dusts, mixtures of gases, the most effective ventilation practice, and so forth. Most exacting procedures have been developed in government, university and industrial laboratories to improve safety and efficiency in mines.

Among the research projects that have been most stimulating are those conducted by Professor Bucky at School of Mines, Columbia University. Some of these have been research projects of the Engineering Foundation and Columbia University. In general, these have been studies of mining problems in the laboratory, the objective being to determine the nature of the forces developed during mining operations, the behavior of specific types of rocks and ores when subjected to such forces and the determination of appropriate dimensions of mine openings, pillars and so on, to secure the desired results. The tools used are "barodynamic centrifuges, photo-elastic apparatus and the mathematical and physical sciences. Work at the mining laboratories at Columbia has resulted in contributions to the theory of block caving, and stress-distribution in ore bodies, pillars, roofs and to drawing practice."

Some of the institutions have offered graduate courses in geology for a number of years, while graduate work in geophysics at several schools has been offered for at least 15 years.

In petroleum production and refining, graduate instruction has been offered at several of the institutions having accredited undergraduate curricula in petroleum engineering. Professor Uren presented a petroleum engineering curriculum for graduate work, following a four-year undergraduate curriculum and possibly a period of field experience.[14] He said:

> Engineering schools and colleges catering to this need (for graduate work) should offer advanced courses affording intensive and detailed treatment of groups of subjects appropriate for graduate specializations. Advanced petroleum engineering courses may be directed toward a thorough understanding of the intricacies of reservoir mechanics, or various aspects of drilling and production technique. The research and development specialization may include advanced courses in physics and chemistry, with others affording training in research methods. The oil-field exploration specialization affords an opportunity to explore advanced phases of structural geology, paleontology, sedimentation and lithology of reservoir rocks, geophysics, and related subjects.

Reference has been made previously to the graduate work and research in ceramics at schools for mineral industry education, particularly the New York State Ceramic Experiment Station.

Research that has been carried on in connection with college instruction has at times taken the form of projects developed by some outstanding member of the teaching staff, and at times has been part of the work of advanced undergraduates and graduate students. More recently, much of the research work has been coordinated and directed through a department of research, an experiment station, or other division of the institution. This work has been financed in various ways, sometimes by endowment, sometimes by special appropriations of state legislatures and sometimes by funds supplied by industrial corporations or associations. Mineral Industry Experiment Stations have been established at various educational institutions, one of the first being the Missouri State Mining Experiment Station which was established in 1909.

The Colorado School of Mines experimental metallurgical and ore-dressing plant was established in 1912 to provide laboratory facilities for student training in the fundamentals underlying metallurgical processes and to furnish equipment and space for research work in ore treatment, both theoretical and applied. The stated objects of the theoretical research have been to extend present theories and to correlate and coordinate theory with practice. The staff of the plant has cooperated with industrial organizations by means of fellowships and otherwise in all kinds of investigations with regard to ore treatment.

The United States Bureau of Mines has maintained a field station on the campus at Golden where investigations on utilization of western fuels are conducted by the Coal Division of the Bureau. Unit processes are investigated with the assistance of mechanics and skilled laborers and with the part-time help of students who attend the Colorado School of Mines.

One of the most comprehensive programs in mineral industry research is that organized at Pennsylvania State College in the Mining Experiment Station which was established in 1919. Industrial research is in progress on a great variety of problems of the mineral industry.

A cooperative agreement was entered into June 1, 1919, between the United States Bureau of Mines and Carnegie Institute of Technology whereby the mining and metallurgical industries in Western Pennsylvania could take full advantage of the laboratories, equipment and library of the United States Bureau of Mines at Pittsburgh, as well as of the advice and instruction of its technical staff for carrying on educational work and research fellowships in mining, utilization of fuels and

metallurgy. The Carnegie Institute extended to employees of the Bureau the privilege of taking courses at the Institute under the same conditions as to fees as are extended to members of the Institute faculty. Cooperative mining and metallurgical studies through research fellowship were initiated in 1920. Advisory Boards in metallurgy and in mining determined what problems should be undertaken and published a number of reports, after editing by the Bureau of Mines. Eight or more fellowships paying $750 per year for ten months were established each year. These fellowships were open to qualified graduates of universities and technical schools. The fellows were required to register at the Institute as graduate students and become candidates for advanced degrees. Under this agreement, a total of 75 bulletins were published between 1922 and 1937. The work was discontinued in 1939.

This research fellowship plan of the Bureau of Mines was extended to the University of Washington, University of Arizona, University of Minnesota, Missouri School of Mines and the University of Alabama. Dr. A. C. Fieldner advises that this educational program proved quite successful in training men in advanced and specialized work in the mining and metallurgy industries. He states:

> The fellowship system has declined in the last decade because of lack of money, in some instances, and also because in many universities their own research foundations were established. In such instances, they preferred to limit their graduate instruction to men who were taking research in a department of the university itself rather than the neighboring institutions of the Bureau of Mines Experiment Stations. At the present time, Bureau of Mines fellowships are maintained at the University of Alabama and the University of Washington. These fellowships are engaged on problems of coal beneficiation or ore dressing.

At the present time graduate work in industry is being carried on by several technical institutions in affiliation with industrial corporations. The University of Pittsburgh is conducting graduate work in the mineral industry in metallurgical engineering, petroleum engineering and geology. Certain industries in the Pittsburgh area provide expert instruction in special fields by competent employees and the University of Pittsburgh has arranged with these affiliated companies to appoint these "Industrial Lecturers" to the graduate faculty of the University. This arrangement permits the proper coordination and supervision of the work by the University and the granting of graduate credit.

This graduate work in industry at Pittsburgh has been done in affiliation with the Westinghouse Electric Corp., the Aluminum Company of America, the Philadelphia Company and subsidiary companies, the Carnegie-Illinois Steel Corp., and the Gulf Research and Development Co. The graduate work of the Carnegie-Illinois Steel Corp. has been developed also at Chicago and Gary in affiliation with the Illinois Institute of Technology and Indiana University.

In addition to the courses given by the "Industrial Lecturers," graduate students who are candidates for advanced degrees must complete other courses given by faculty members in accordance with the standard requirements of the educational institutions.

The course material presented by the "Industrial Lecturers" has been prepared by a staff of experts of the several corporations; the material has not been publicized and probably is not equaled in any text available for other types of graduate study. The text of the lectures prepared by the staff of the Carnegie-Illinois Steel Corp. surpasses anything the writer has ever seen in the form of instructional material for undergraduate and graduate students.

Reference is made later to the training of men by the United States Geological

Survey and other scientific and technical organizations. Mention should be made also of the work being done in the great industrial research laboratories in training men who eventually find places in research or production in industry. It is the practice of a number of these organizations to employ promising graduates of schools of mineral industry, train them in the specific research work assigned to them and make it possible for them to pursue graduate-laboratory work in colleges and universities in or nearby the industrial center. The long-range effect of such programs has been most constructive.

Reference has been made previously to the Cooperative Research Program that was carried on for a number of years at Carnegie Institute of Technology (Pittsburgh, Pennsylvania). In 1934 a Bureau of Metallurgical Research was organized for work primarily in physical metallurgy. Since 1935 this research work has been carried on as the Metals Research Laboratory of the Institute.

In 1931 the Coal-research Laboratory was established at Carnegie to do fundamental research on coal.

A most valuable program in the field of research education has been that of Battelle Memorial Institute (Columbus, Ohio), which in 1931 organized work for graduate research fellows and postdoctoral research associates. According to the announced plan, associates and fellows are brought together for a year's "interneship" at Battelle for the purpose of developing highly trained research men, primarily for careers in industrial research. Appointees devote their full time to their own research projects in the Battelle laboratories under the guidance of the Battelle technical staff. The projects must be of a fundamental or general nature, leading to the discovery of scientific principles or the gathering of significant new data. Among the fields listed as appropriate for such research are: fuels and combustion, ceramic engineering, mineral preparation, metallurgy-physical (process, ferrous, nonferrous), high-temperature alloys (welding) foundry technology and production research.

These fellows participate in some of the seminars for study in any field related to research. This training program is designed to advance the cause of research in general rather than as a source of men for Battelle.

The Engineering College Research Association has recently become the Engineering Research Council of the American Society for Engineering Education. Its general purpose is to assist in developing the research facilities of engineering colleges. Among the functions proposed is collaboration with various existing agencies to prevent duplication of effort and achieve maximum utilization, coordination and development of engineering and scientific research facilities.

Post-collegiate Education

The "power of growth" has been referred to as one of the most essential qualifications of an engineer. The graduate of a four-year course who enters employment in the mineral industry is just starting his education.

Evan Just, Editor of *Engineering and Mining Journal*, has well said:

> Today education never stops, college training is only a very small beginning. The alert engineer must keep abreast not only by such direct experience as he is able to obtain, but by constant perusal of technical publications and attendance at meetings of various mining societies where technical papers are presented.

Through the years a number of the companies who have employed graduates of four-year courses have endeavored to train them for efficient work in the industry. This has been accomplished largely by having job-training or other scheduled programs which permitted the young man to spend some time working at different jobs

or in different departments in order to give him some knowledge of the various classes of work and the operation as a whole.

One of the first mining companies to organize work of this character was the Hudson Coal Co.[15] This work was started in 1915 and elaborated in 1925. In 1925, direction of this work was placed in charge of one man who spent his entire time looking after the students, examining their work, reading and criticising reports and helping them in their studies which they were encouraged to continue. From 1925 to 1929 eighteen college graduates and sixty men who had been employed previously by the company finished the special courses. Most of them were placed in supervisory positions. During the depression it was necessary to give up this complete and thoroughly satisfactory program. In recent correspondence, Mr. Cadwallader Evans, Jr., President of the Hudson Coal Co., said:

> We have recently reinstated courses on not quite so extensive a basis as formerly. We now give the men whom we select special training of about a year. When we had a good supply of trained men we made it a rule not to promote a man to a responsible position until he had received some special training.

It is believed generally that Mr. Evans has made a most valuable contribution in the field of training engineers following their graduation from college. The older men he has trained occupy important places in the mining industry and the men trained in more recent years give promise of rendering most valuable service as leaders in responsible positions.

The "post-education" program of the Pittsburgh Coal Co. was described by Mr. H. R. Wheeler in 1940.[16]

> The training program was divided subjectively into eight sections; an introductory period, underground mining operations, coal preparation, inspection and mechanical maintenance, engineering, industrial relations, operating analyses and foremen's training. Approximately 120 weeks are required for the student to complete the course. The same training program is applicable to both graduates and undergraduates, but undergraduates who work with the organization during their summer vacation will complete a portion of the program before graduation from college.

During the introductory period, which extended over one week, the activities, policies and objectives of the company were explained. At the end of the week men were assigned to labor jobs at the mines for a period of 25 weeks. Before leaving the mine, each man spent two weeks as an observer of the assistant foreman and fire boss. Upon the completion of the underground-labor period, he was transferred to a preparation plant where he worked 24 weeks as a laborer and a laboratory and sampler's assistant. He was then employed for 31 weeks as an apprentice in the mechanical-maintenance shop and as an assistant to the various company inspectors, including safety, mechanical equipment, lubrication, preparation and fire prevention. He then spent 16 weeks in the engineering department to familiarize himself with mine layout and projection. Following this he was assigned to the industrial-relations department for 12 weeks as time-study engineer, clerk in the central employment office and student of the union contract and case histories of labor disputes. Finally, he was assigned to an assistant production manager under whose direction he worked on special assignments, such as analyses of operating costs, compilation of operating statistics, study and development of operating problems and experiments.

Mr. Wheeler said that the program was directed toward the development of operating men, as it was in operating work that the college graduate was most needed and where the greatest opportunity existed for the greatest number of men.

The results of this work were presented

by Mr. J. E. Norton in 1946 in a publication of the Institute.[17] He said future plans are unsettled as to some details, but as it is felt the industry needs technically trained men, they plan to follow the same general training program, including summer work for undergraduates. The program schedules wage increases every six months for two years, provided the men show promise. If a man does not show promise, he is dropped or told he is on trial and the reasons explained to him.

One of the largest of the metal mining companies has one schedule for graduates desiring to follow underground operations and another for men planning to follow technical work. The man planning to follow operating must have at least six months practical mining experience in order to be eligible for operating work in the mining department. For a technical career, the graduate need not spend the six months underground but must serve an appropriate apprenticeship in the several technical departments. Graduates beginning work in the metallurgical operations usually go through the metallurgical research department after gaining some experience in practical operating, from where they are advanced to assistant superintendent in various operating departments.

In discussing post-collegiate education of mining engineers, Dr. T. T. Read suggested that there is a need in the metal-mining industry for organized and planned meetings of young technical employees for presentation and discussion of technical problems of their own work. This would serve to educate them to analyze and present problems. Such discussions would give them a consciousness that they are developing professionally.[18]

While many of the larger corporations are prepared to train college graduates entering their employ and some prefer to give such training, there are many smaller companies that do not have the facilities to give such training. One of the results is that after young men have had the training afforded by the larger companies, they are persuaded frequently to join the staff of employers who have not organized post-collegiate training.

As long as many employers in the mineral industry expect the graduate of a four-year course to be able to start work without much supervision, it will be necessary for technical colleges to give, at least as optional courses, some of the training required for service in industry.

Subcollegiate and Vocational Education

Subcollegiate instruction has been reviewed very completely by Dr. Read in Chapter 14 of *Development of Mineral Industry Education.* In the United States over 50 institutions of collegiate or university rank are offering subcollegiate work for the mineral industry. In 1919, Professor A. C. Callen made a survey of subcollegiate education for the mineral industries for the Federal Board for Vocational Education. He called attention to the fact that probably the first industrial training in the United States for mine workers was that organized by Eckley B. Coxe for coal miners at Drifton, Pennsylvania, in 1879. Professor Callen classified subcollegiate instruction for the mineral industry as follows: mining schools, lecture courses, correspondence courses, short courses in mining colleges, industrial mining schools and underground schools.

A great stimulus was given to education of this type by the requirements of mining laws in the various coal-mining states. The work of the United States Bureau of Mines has played an important part in all phases of subcollegiate education, particularly in first-aid, mine rescue, and safety.

The need for vocational training in mining has been recognized in the metal-mining field as well as in coal mining. Vocational

schools were established in Nevada in 1903. The Copper Queen Mining Co. began an evening course for miners in 1918. The same year in cooperation with the State Board of Vocational Education and the Federal Mining and Smelting Co., an underground school for training of mining labor was established in the Coeur d'Alene district under the direction of Dr. F. A. Thomson. Annual mining institutes have been held under the auspices of the University of Washington for a period of 20 years. Because of a shortage of coal miners, a training school was established underground at the Stag Canon Coal mining branch of the Phelps Dodge Corp. in New Mexico.

The Anaconda Copper Mining Co. maintains a department for the training of new men starting work underground. The new man is trained at a school on the surface where he is made familiar with mining operations, mining machines and tools. Having been given this preliminary training, he is assigned to a "student stope" where he works with more experienced students and takes his regular turn at various mining operations. He works under the supervision of a boss who receives directions from an instructor who handles several stopes. All instructions are based on the principles of showing the man how to do the work and making sure that he has learned how. The period of training varies with each individual case, but the average is about six weeks. At the end of the training period the new man is placed in a regular stope with experienced miners, but his training and work are followed up by shift bosses. The subject matter of the training course comes under four main headings: safety, mining methods, human relations and service.

The permanent government organization for vocational instruction in mineral industry work is that of the Vocational Division of the Office of Education, which originally was organized as a separate Federal Board of Vocational Education in 1917. Details regarding this program are summarized by Dr. Read. Much of the present extension work, noted later, is organized under the Smith-Hughes Act and subsequent legislation.

Among the most effective examples of mining extension work are the courses conducted by West Virginia University and by Pennsylvania State College. The work in West Virginia was organized in 1913. In 1915 there was an enrollment of 220, and in 1916, 1806 were enrolled. At the present time there are nine full-time and several part-time instructors throughout the State of West Virginia at 51 centers, and some 2000 to 2200 students are enrolled each year. A six-week summer course is scheduled.

The extension work at Pennsylvania State College is on a large scale. In 1939 to 1940, the registration was 3964 in the 173 classes in 99 class-centers in the state. The Extension Division has published 15 textbooks for the use of the extension students. Courses are given in coal mining, ferrous metallurgy, ceramics, petroleum and natural gas.

Ohio State University is conducting work at ten centers in Ohio with 250 students enrolled.

Mention should be made of the training of young men enrolled in vocational mining courses at high schools. In a paper by D. C. Jones, Supervisor of Mining Extension, Pennsylvania State College, entitled *The Development of Future Miners through High School Training Programs*, he states that high schools or vocational trade schools are offering instruction in coal mining as follows: Kentucky (1), Ohio (1), Pennsylvania (3) and West Virginia (6). The work in West Virginia is proving very successful.[19]

The first school to offer this type of work was the Mapletown High School in Greene County, Pennsylvania, beginning in 1939 to 1940. In 1945 to 1946 there were 1330

students enrolled in eight high schools.

There is a great interest being shown in this type of work in the coal fields and the institutions of higher learning are giving active support to the development of the mining classes in high schools. These vocational courses are supported by the State Department of Public Instruction and receive Federal Aid.

Additional Contributions by Geological Surveys, Mining Departments, Technical Societies and Technical Press

In addition to the work of institutions established specifically for educational purposes, much has been accomplished toward the promotion of formal education and the dissemination of knowledge of the mineral resources and of the mineral industry by national and state geological surveys, departments of mines, national and local technical institutes and societies and the technical press.

This work has been done chiefly by the publication of technical literature, the sponsoring of educational and research programs, and the support of educational work by scholarships, fellowships, loans and so on. Some of the latter activities will be discussed in more detail later.

The teachers in the early days of mining education in the United States and their students labored under a severe handicap as there were few, if any, textbooks available on the technical subjects. Much of the instruction was given by lectures and there was a serious waste of the students' time, as compared with the present-day opportunity of working with well-prepared textbooks. The struggle with German and French texts was not worth the effort for the American student planning to go into American mining fields. As a result of this great need, there soon appeared authoritative textbooks, manuals, and handbooks in science and technology prepared by American teachers, scientists, and engineers. Many of these are classics in their field and have contributed largely to the efficiency of American engineering education. Some of this material has been provided in the form of textbooks prepared by members of the teaching staff of the colleges and universities, and some in bulletins and reports of federal and state geological surveys, departments of mines, scientific and industrial bureaus, and, in some instances, by manufacturers.

When the United States Geological Survey was organized in 1879, it was not planned as an educational institution, but it has proved to be a most effective agency along three lines. First, in providing splendid material for instructional purposes; second, by employing students as field assistants; and third, by employing college-trained men following their graduation and becoming a graduate training school for many of the men who later are selected to teach geology in the schools of mineral industry.

This outstanding contribution of the Survey through the years has been so constructive that a concise statement relative to this work, prepared by an officer of the Survey, is quoted:

> The Survey has not participated directly in the field of formal education, but it has indirectly made large contributions to progress in education through its publications and through the opportunities it has afforded men bent on careers in the mineral industry to obtain training and practice in the practical aspects of the subject. Many of the leading engineers and geologists in the industry are alumni, so to speak, of the United States Geological Survey, having received their early training, in part at least, as members of Survey field parties.
>
> Publications of the United States Geological Survey have been used widely as texts or as standard reference books for advanced classes. All are comprehensive summaries of basic data in fundamental fields of geology.

Universities also make use of many United States Geological Survey publications as collateral reading for advanced undergraduate courses in all branches of geology. Reports of a more detailed nature are utilized, of course, as a source of data for student reports and research. Moreover, as demonstrated by footnotes and reference lists, publications of the United States Geological Survey furnish many of the data and illustrations that appear in texts written by educators. A very considerable number of persons who now are teaching in our universities and colleges have spent from one to several years as members of the Geological Survey staff.

Topographic and geologic maps and folios published by the United States Geological Survey are used widely in colleges, serving as the laboratory tools for instruction in map interpretation, physiography, geologic structure, underground extensions of mineral deposits, etc. In this respect, the Survey has made up sets of topographic maps, illustrating various physiographic types, for use in laboratory work in elementary geology.

It has long been the policy of the Survey to employ students during the field seasons in various capacities as temporary assistants to field parties. This is very consciously an educational policy and one that we hope to continue indefinitely, for it pays dividends, not only to the student by way of giving him a foretaste of the life and work of a geologist or mining engineer, but to the Survey by way of providing high-grade and interested assistants and an opportunity for our officers to become well-acquainted with the young men who are preparing for life work in the mineral industry.

The United States Bureau of Mines was formally established in 1910 and took over from the United States Geological Survey the technologic work which was inaugurated in 1904. It was organized to prevent, if possible, the disasters in coal mines and to reduce the waste of life and of resources in the varied mining and metallurgical industries, and to increase health, safety, economy, and efficiency in mining, quarrying, metallurgical and miscellaneous mineral industries of the country. The first work was directed toward increasing the knowledge of fuels and reducing mine accidents. A great many publications of the Bureau have served most effectively as college textbooks. Many of them were prepared to acquaint employees and employers with procedures to improve safety and health. An extensive collection of educational motion-picture films has been compiled by the Bureau and these films are available for instruction at schools of the mineral industry. These films, numbering more than 10,000, together with those supplied by various manufacturers, are a splendid aid to classroom instruction. Some of the mining schools include in their courses the training in first-aid and mine-rescue given by the Bureau. Thus, in various important ways, the Bureau of Mines is a valuable and effective agency in mineral industry education.

It is necessary to do no more than mention the great contribution of the Institute itself in the preparation and distribution of literature suitable for teaching purposes for it has not only sponsored many timely projects but has made financial arrangements for mining students to receive some of the literature free or at a nominal charge.

The work of the Coal Division of the Institute to promote student interest in coal mining was reviewed by the Chairman of the Committee for Promotion of Student Interest in Coal Mining, Mr. George H. Deike, in the February 1946 issue of *Mining and Metallurgy*. The Committee, recognizing the need for additional technically-trained men in the coal mining industry, has taken steps to develop new interest in this field and has organized programs for scholarships at the various schools for mineral industry education offering courses of instruction in coal mining, fuel technology, and the like.

Under the auspices of the Coal Division of the Institute, a plan was suggested to the National Coal Association for coopera-

tion in the program of securing more technically-trained men and particularly that the National Coal Association finance a full-time co-ordinator whose duty it would be to interest coal producers and associations in sponsoring the better young men from the mining towns in (1) studying engineering and (2) working for the companies during the summer. The co-ordinator would develop the interest of the coal industry in young men, and vice versa, and build up a supply of men with the best type of educational background for careers in the coal industry.

As a result of these efforts, the National Coal Association approved the proposed program and appointed Mr. M. D. Cooper as Manager of Vocational Training. Mr. Cooper entered upon this work on January 1, 1947.

In reviewing the work of the Mining and Metallurgical Society of America, in the field of education, Dr. Read[20] refers to the report of a committee on technical education published on September 30, 1921. The report dealt largely with the content of curricula and a general discussion of mining education. In 1921, a committee was appointed on vocational training for metal mines and a report was submitted describing the aims and specific procedures of projects then active. Other American technical and scientific organizations have made important contributions to mineral industry education.

A number of manufacturers have prepared comprehensive handbooks for reference purposes and these, too, together with bulletins on special subjects have made available the latest information and data on machinery, equipment and practice in the operation of such machinery and equipment.

Soon after the organization of the Institute, it was announced that arrangement had been made for the *Engineering and Mining Journal* to become the "organ" of the Institute for the first publication of proceedings, papers and notices to members for one year. Founded on March 31, 1866, as the American Journal of Mining, this technical journal has continued through the years as an important and effective institution in disseminating technical information and in cooperating with sound and enterprising programs in mineral industry education. Other substantial and progressive technical periodicals have given effective assistance in educational work, both by publishing currently important technical knowledge and by issuing comprehensive volumes on mining practice.

Endowments, Gifts and Aids to Students

Substantial assistance has been given to mineral industry education by the establishment of endowments for the purpose of providing scholarships, fellowships, conducting research, publishing technical literature and making technical literature available through the support of the library of the engineering societies.

The James Douglas Library Fund started in 1919 is used for the support of the Library. The Seeley W. Mudd Memorial Fund was established for the encouragement of research, the dissemination of knowledge and for the promotion of the welfare of engineers engaged in the professions of mining and metallurgy. Some of the proceeds of the Rocky Mountain Fund has been used to publish technical volumes.

The Hayden Foundation has for its purpose the promotion of the interests and welfare of youth and some of the proceeds has been used for the technical education of the Student Associates through the preparation and circulation of technical publications.

A volume on *Petroleum Conservation* is to be published with the proceeds of the Henry L. Doherty Memorial Fund, as well

as the 1946 volume of Petroleum Production Symposium (Statistics).

Endowment Fund *X* was created to support the scientific, literary, or educational purposes of the Institute. The Robert C. Gemmell Memorial Fund was established primarily for educational purposes and undertakings that will tend particularly to give aid to younger members of the profession. The Anthony F. Lucas Fund was established to promote the welfare of the Petroleum Division, including the granting of scholarships.

In addition to prizes for papers written by students, there are awards for outstanding papers prepared by younger members of the Institute. These prizes and awards are made possible by endowments and by current donations by individual members, by Local Sections, and by Divisions.

In discussing the improvement of facilities for instruction, Dr. T. T. Read has pointed out[21] that a number of college buildings were erected during the closing years of the nineteenth century and the first years of the twentieth, some of them being the gifts of individuals. In 1903 John Hays Hammond gave a mining and metallurgical laboratory to Yale; Simon Guggenheim gave a building to the Colorado School of Mines; Adolph Lewissohn gave a School of Mines building to Columbia. Mrs. Hearst was interested in the building program of the University of California and, as a memorial to her husband, gave a College of Mining building in 1907. The Mackay family gave the University of Nevada a School of Mines building in 1906. John D. Rockefeller gave funds for a mines building at Case Institute of Technology in 1906.

John Markle made provision for a mining and metallurgy building and the support of mineral industry instruction at Lafayette College. In 1940 the Phelps Dodge Corp., as a memorial to James Douglas, provided a large sum for erecting and equipping a mining building for the University of Arizona.

All of the other important buildings that have been erected in recent years for mineral industry education have been provided from public funds, including buildings at Montana School of Mines, Colorado School of Mines, Pennsylvania State College, West Virginia University, Missouri School of Mines and South Dakota School of Mines.

In commenting on the gift of buildings for mineral industry education, Dr. Read said:

> In some cases the buildings have imposed on the institutions to which they were given a necessity for securing continuing income to maintain a staff to utilize these physical resources. This in turn leads to emphasis on undergraduate work which attracts a greater number of students, and is less expensive to maintain than graduate work. Advanced study, the peculiar province of institutions of higher learning, thus tends to be neglected, unless it can be correlated closely with current commercial problems.

It should be noted, however, that in December 1945, the Kennecott Copper Corp. made a gift of $250,000 to the University of Utah for the School of Mines. This fund is not to be used for buildings, but for the support of research and education. Among the subjects proposed for research were the treatment of low-grade ores and methods of geophysical prospecting. At the present time the mining department is conducting research on shaped explosive charges.

Trends in Progress and Summary

The broadening of the scope of the mineral industry has brought about changes in the programs and the organization plans of some of the educational institutions. In 1929 the School of Mines of Pennsylvania State College announced a reorganization

of its mining and metallurgical curricula and adopted the name of School of Mineral Industries. Dean Steidle has pointed out that the School is organized to include the broadest application of earth sciences, the extraction of mineral wealth, and mineral processing. The school offers curricula leading to the B.S. degree in (1) mining engineering, (2) metallurgy, (3) earth sciences, (4) ceramics, (5) petroleum and natural gas engineering and (6) fuel technology.

It has been announced recently that the new School of Mineral Industries of the University of Utah will be organized along somewhat the same lines as Pennsylvania State College. Stanford University has announced the establishment of a new School of Mineral Sciences, combining the Departments of Geology and Mining.

Among the points that indicate progress in education is the increased interest in the development of the individual student. While considerable thought has always been given to the training and development of the individual, there is now more of a directed effort to reach all of the men in the group that enroll in the mining courses. Probably there has been more attention given to helping the student of average ability.

About 1913, one of the greatest of American educators, in a commencement address, said that in his opinion the prime function of a university was to develop the "intellectual mountain peaks." He pointed out that the great advances in civilization had been the result of the contributions made by these so-called "intellectual mountain peaks." In a recent address, Herbert Hoover discussed "The Need for Uncommon Men and Women," and said:

"If we are to have leadership in government, in science, in education, in the professions, and in the home, we must find and train some uncommon men and women. Let us remember that the great advances have not been brought about by mediocre men and women. Rather they were brought about by distinctly uncommon men and women with vital sparks of leadership. Many of these great leaders were, it is true, of humble origin, but that was not their greatness. Our sure hope of recovery in the moral and spiritual world is the wealth of uncommon men and women among our people. And it is our educational institutions that will promote and train them."

While no one disputes the fact that contributions have been made by these great intellects and that progress generally depends on such contributions, the fact remains that the application of these great scientific discoveries depends on the intelligent and devoted service of outstanding men who have creative and constructive ability of no small caliber, even though they are not "intellectual mountain peaks."

Probably all will agree that the mineral industry needs "uncommon men" and "intellectual mountain peaks." It has been the policy of the officers and teachers in mineral industry education to try to discover the men with unusual talents during their four years of undergraduate instruction and to see that they have the opportunity to continue their training in graduate work and research. This has been done without sacrificing the quality of work for the group of undergraduates who take only the four-year curricula. The Engineers' Council for Professional Development is to be commended for insisting on sound training in engineering fundamentals for all candidates for degrees in all curricula in mineral industry education. The mineral industry is interested in graduate study and research but progress in these fields cannot be made unless the undergraduate is thoroughly grounded in the fundamentals of engineering.

Recently, Mr. L. W. Chubb, Director of the Westinghouse Research Laboratories, called attention to certain matters that deserve most careful consideration of engineering educators.[22]

"Today, in the esteem and appreciation of the public, science is in the saddle. The most glamorous accomplishments, such as radar, the atomic bomb, and the proximity fuse, are applications of scientific knowledge recently acquired. They are accredited mostly to scientists, particularly chemists and physicists, who in special groups actually carried on the development. Probably over 95 per cent of the activity on these glamorous items consisted of engineering and production. All of them were applications of pre-war scientific knowledge. Although a great amount of scientific research had to be done, most of the work of the scientists was involved in engineering development, and these are activities quite outside of their usual field."

Substantial progress is being made in graduate work and in research at various colleges and schools offering curricula for mineral industry education. It is important that undergraduate instruction be maintained at the highest standards. If sound engineering instruction is not maintained at these institutions, the American mining industry will suffer greatly. Fortunately, in most schools for mineral industry education the high standards of engineering curricula have been maintained. Graduate work and research in both science and engineering should be conducted but not at the sacrifice of undergraduate engineering instruction.

In the introductory statement, mention was made of a number of the most significant points that are evident in a review of the progress of mineral education since the founding of the Institute. Among the most important of the trends observed are:

1. Maintenance of high standards of education in fundamental sciences and engineering, with improved textbooks, laboratories and plants.

2. Increasing interest of leaders in the mineral industry in the work of the educational institutions and other agencies for training men for the industry.

3. Adapting and developing curricula and courses of instruction to meet the needs of a progressive industry.

4. Active effort to select students best suited for mineral industry education and counseling them during college years and the years immediately after graduation.

5. Increasing interest in graduate work and research.

The efficient members of the teaching staff of the schools of mineral industry education have been doing splendid work in American mineral industry education, not only in formal instruction but in developing standards and ideals. Much of this work has been coordinated and guided by the American Society for Engineering Education and the Engineers' Council for Professional Development. The American Institute of Mining and Metallurgical Engineers is committed to educational progress, not only through the colleges and technical schools providing undergraduate instruction, but throughout the entire profession and the mineral industry.

Financial support for various purposes has been commendable and there appears to be a greater realization among the leaders of the industry and the major corporations that the future of the mineral industry in the United States depends largely on the progress of the schools and in training men. There has been increasing financial support for research, general maintenance and student aid.

In addition to the institutions organized specifically to offer instruction of collegiate grade, there have been a number of other agencies that have made valuable and continuing contributions through the years in investigations, research and technical publications. These include the national and local engineering institutes and societies, the federal and state geological surveys and mining departments, the technical press and research and training departments of industrial corporations and research laboratories. A great fund of valuable technical

information has been made available for teaching purposes.

The educators comprising this Division have followed many inspiring and efficient teachers in the schools and colleges of the American mineral industry; they have improved practices and raised standards that were high. The mining fraternity honors them for the progress made in a world that looks to science and engineering for leadership.

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Some Postwar Problems in Geological Engineering Education

By W. T. Thom, Jr.,* Member AIME

(New York Meeting, February 1948)

All engineering education is faced by certain basic problems, three of which seem to have particular present importance in geological engineering training in general, and in respect to training for oil field geoexploratory work in particular.

Problem 1

The first of these problems has to do with present and future educational objectives and with the numbers and qualities of the men who should be coming out of our geoexploratory training programs. Before the war we needed ever-increasing numbers of graduates at the Bachelor's level and a very much smaller number of men who had reached the Doctorate degree. Now, with more difficult, more extensive, and more costly exploratory work; with a rapid expansion of company "staff" activities, and with the initiation of many new research organizations and laboratories; it is clear that men of exceptional ability will be more and more in demand, and that the schools will need to carry a greater *proportion* of their students through to the Ph.D. degree.

Problem 2

Assuming that the mineral industries *will* need an increasing supply of well-qualified Ph.D.s for future geoexploratory work (staff, line and research), can the schools expect to provide this supply of able Ph.D.s merely by expanding the proportion of B.S. and B.A. majors who go on through to the doctorate stage of training? And the answer is definitely "No"—as the situation stands at present. Most A.B. graduates are years short of the preparations in basic science and elementary engineering prerequisite to effective graduate study in the geoexploratory fields (paleontology excepted), and similarly many of the annual crop of B.S. graduates lack adequate training in English; in ancient and modern history; and in the modern principles of labor and public relations. Indeed, *several* basic problems of undergraduate curricular organization and integration remain to be solved before *most* recipients of a Bachelor's degree will have that sound and broad educational foundation prerequisite to graduate training on a really effective basis.

It is for these reasons that we have continued to experiment with our geological engineering program at Princeton, in an endeavor to provide a pilot-plant demonstration of a suitably broad and flexible undergraduate curriculum.

Problem 3

But assuming that the schools can and do get undergraduate curricula set up in proper phase and scale to meet needed enrollment quotas for graduate training, what about the *most* urgent part of our whole mineral-industry educational problem? What about *finding* and *bringing into* mineral-industry training those youngsters of superior ability who are, by potential or known inclination, especially well-qualified for geoexploratory work?

Manuscript received at the office of the Institute Feb. 15, 1948. Issued as TP 2493 in Mining Technology, November 1948.

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At present we are *not* making contact with most of these boys at the optimum time for the awakening of their latent interests, and for their orientation and guidance. This optimum time for making contact comes before or during the summer after junior year of High School, at which time a youngster arrives at an important road-fork in his choice of objectives—where a deflection in the wrong direction puts him under a heavy, perhaps insuperable, professional handicap.

The task of finding and orienting the able High School students who are "naturals" for mineral-industry work seems to be definitely within the function and responsibility of the AIME as a professional (and educational) organization. Furthermore, under the sponsorship of such a major professional society, it should be possible to keep the orientation and advising of able High School students either from becoming enmeshed in inter-departmental or inter-school rivalries; or from being hopelessly distorted by the publicity currently being given to the "glamor areas" of science, such as nuclear physics.

Summary

To recapitulate, the three big questions in geological engineering education, as in all engineering and mineral industry training, are:

1. What are the indispensable numbers of qualified graduates that we need to train both to the Bachelor's degree and to the Doctorate?

2. Are our educational "mills" properly equipped and coordinated to supply the present and future demands for high quality graduates?

3. Are we, as members of the AIME, doing the research and development work that is needed if we are to ensure and assure the mineral industries (and the nation), through attraction of our fair share of the able High School graduates into our geoexploratory and general mineral industry training programs? We of the Mineral Industries should not seek to divert good physicists from being physicists—but neither should we allow good mineral-industry engineers to become mediocre physicists, merely for want of a few timely words of enlightment and advice.

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(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of papers in *italics*.)

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